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Baptiste Nickel Project

NI 43-101 Technical Report and Prefeasibility Study

British Columbia, Canada

Effective Date: September 6, 2023

Report Date: October 18, 2023

Prepared for: FPX Nickel Corp.

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Kevin Murray, P. Eng., Ausenco Engineering Canada Inc.
Ronald Jaap Voordouw, P. Geo., Equity Explorations Ltd.
Jeffrey B. Austin, P. Eng., International Metallurgical & Environmental Inc.
Cristian H. Garcia Jimenez, P. Eng., TechSer Mining Consultants Ltd.
Duke Barret Reimer, P. Eng., Knight Piésold Ltd.
Richard FJ Flynn, P. Geo., Next Mine Consulting Ltd.
David John Baldwin, P. Eng., Carisbrooke Consulting Ltd.
Paul Mysak, P. Eng., Onsite Engineering Ltd.
Harold Rolf Schmitt, P. Geo., ERM Consultants Canada Ltd.

CERTIFICATE OF QUALIFIED PERSON
Kevin Murray, P. Eng.

I, Kevin Murray, P.Eng., certify that I am employed as a Manager Process Engineering with Ausenco Engineering Canada Inc. ("Ausenco"), with an office address of 1050 West Pender Street, Suite 1200, Vancouver, British Columbia, Canada, V6E 3S7.

1. This certificate applies to the technical report titled "Baptiste Nickel Project NI 43-101 Technical Report and Prefeasibility Study" that has an effective date of September 6, 2023 and a report date of October 18, 2023 (the "Technical Report").
2. I graduated from the University of New Brunswick, Fredericton NB, in 1995 with a Bachelor of Science in Chemical Engineering.
3. I am a member of the Association of Professional Engineers and Geoscientists of the Province of British Columbia, Registration number# 32350 and the Northwest Territories Association of Professional Engineers and Geoscientists' Registration# L4940.
4. I have practiced my profession for 23 years. I have been directly involved in all levels of engineering studies from preliminary economic assessments (PEAs) to feasibility studies. I have lead preliminary test work design, test work analysis and flowsheet development as well involvement in detailed design and commissioning. I have also developed operating cost estimates and contributed to and reviewed capital cost estimates. I have 14 years of direct experience in testwork and design and cost estimations for copper and nickel hydrometallurgical processes including pressure oxidation, solvent extraction and electrowinning, impurity removal and crystallization as well as detail design and commissioning of a copper concentrate pressure oxidation, solvent extraction and copper electrowinning plant in Brazil.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I visited the Baptiste Nickel Project site on June 27, 2023 for a visit duration of one day.
7. I am responsible for 1.1, 1.13, 1.14.2, 1.14.5, 1.15, 1.17, 1.18, 1.20, 1.21, 1.22.4, 1.22.7, 2, 3.1, 3.3,3.4, 17, 18.1, 18.2.3-18.2.5, 18.3-18.6, 18.9.2, 18.10, 18.11.2,18.11.3, 18.12,18.13, 19, 21.1, 21.2.1, 21.2.2, 21.2.3.1, 21.2.4, 21.2.6, 21.2.7.2, 21.2.8, 21.2.9, 21.3.1, 21.3.2, 21.3.4-21.3.7, 22, 24.1, 24.2.1-24.2.9, 24.2.11,24.2.12, 25.1, 25.8, 25.10-25.13, 25.14.1.3, 25.14.1.6.2-25.14.1.6.3, 25.14.1.7, 25.14.1.9, 25.14.2.3, 25.14.2.7, 26.1, 26.2.4, 26.2.7, 26.3, and 27 of the Technical Report.
8. I am independent of FPX Nickel Corp. as independence is defined in Section 1.5 of NI 43-101.
9. I have had no previous involvement with the Baptiste Nickel Project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 18, 2023.

"Signed and sealed"

Kevin Murray, P. Eng.

Ausenco Engineering Canada Inc. Permit to Practice Number. 1001905, Engineers and Geoscientists British Columbia.

CERTIFICATE OF QUALIFIED PERSON
Ronald Jaap Voordouw, P. Geo.

I, Ronald Voordouw, P.Geo., certify that I am a Partner and Director Geoscience with Equity Exploration Consultants Ltd., with an office address of 1238-200 Granville Street, Vancouver, British Columbia, Canada.

1. This certificate applies to the technical report titled "Baptiste Nickel Project NI 43-101 Technical Report and Prefeasibility Study" that has an effective date of September 6, 2023 and a report date of October 18, 2023 (the "Technical Report").

2. I graduated from the Memorial University of Newfoundland in St John's, Newfoundland, in 2006 with a PhD in Geology and from the University of Calgary in Calgary, Alberta, in 2000 with a Bachelor of Science in Geology.

3. I am a professional geoscientist (P.Geo.) registered with Engineers and Geoscientists of British Columbia (License No. 50515), Professional Engineers and Geoscientists of Newfoundland and Labrador (License No. 06962), and the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (License No. L5245).

4. I have practiced my profession for 17 years in Canada, South Africa, and Brazil, and have working experience on several nickel-focused projects, including a PhD thesis on geology of the Voisey's Bay nickel camp (Labrador), post-doctoral geometallurgical research on the Bushveld UG2 deposit (South Africa), and consulting work on the Decar and Giant Mascot nickel projects in BC.

5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

6. I visited the Baptiste Nickel Project site between 4th and 7th July, 2022, for a visit duration of four days.

7. I am responsible for sections 1.2-1.8, 1.19, 1.22.1, 4-12, 23, 25.2-25.4, 25.14.1.1, 25.14.2.1, 26.2.1, and 27 of the Technical Report.

8. I am independent of FPX Nickel Corp. as independence is defined in Section 1.5 of NI 43-101.

9. I have done consulting work on the Baptiste Nickel Project since 2017 as an author of previous technical reports.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 18, 2023.

"Signed and sealed"

Ronald Voordouw, P. Geo.

Permit to Practice No. 1000183, Engineers and Geoscientists British Columbia.

CERTIFICATE OF QUALIFIED PERSON
Jeffrey B. Austin, P. Eng.

I, Jeffrey B. Austin, P. Eng., certify that I am an independent consultant of International Metallurgical & Environmental Inc., located at 906 Fairway Crescent, Kelowna, British Columbia, and incorporated in 1995.

1. This certificate applies to the technical report titled "Baptiste Nickel Project NI 43-101 Technical Report and Prefeasibility Study" that has an effective date of September 06, 2023 and a report date of October 18, 2023 (the "Technical Report").
2. I graduated from the University of British Columbia in 1984 with a Bachelor of Science degree in Mining and Mineral Process Engineering.
3. I am a member, in good standing, of the Association of Professional Engineers and Geoscientists of British Columbia, license No. 15708.
4. I have practiced my profession continuously for 39 years and have been involved in the design, evaluation and operation of mineral processing facilities during that time. A majority of my professional practice has been the completion of test work and test work supervision related to feasibility and prefeasibility studies of projects involving flotation technologies.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the Baptiste Nickel Project site.
7. I am responsible for 1.9, 1.22.3, 13, 25.5, 25.14.1.2, 25.14.2.2, 26.2.3, and 27 of the Technical Report.
8. I am independent of FPX Nickel Corp. as independence is defined in Section 1.5 of NI 43-101.
9. I have had no previous involvement with the Baptiste Nickel Project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 18, 2023.

"Signed and sealed"

Jeffrey B. Austin, P. Eng.

Permit to Practice No. 1002240, Engineers and Geoscientists British Columbia.

CERTIFICATE OF QUALIFIED PERSON
Cristian H. Garcia Jimenez, P. Eng.

I, Cristian H. Garcia Jimenez, P.Eng., certify that I am employed as a Principal Mining Engineer with TechSer Mining Consultants ("TechSer"), with an office address of 540 Hermosa Ave., North Vancouver, British Columbia, Canada, V7M3C1.

1. This certificate applies to the technical report titled "Baptiste Nickel Project NI 43-101 Technical Report and Prefeasibility Study" that has an effective date of September 06, 2023 and a report date of October 18, 2023 (the "Technical Report").

2. I graduated from Universidad de Santiago de Chile, Santiago, 2001 with a degree in mining engineering.

3. I am a registered Professional Engineer with Engineers and Geoscientists of British Columbia (Registration No. 58399).

4. I have practiced my profession for 20 years since graduation. I have been directly involved in several studies, working in consulting and operations in project evaluation, technical services group and strategic mine planning, mine design and mineral reserve estimates. Experience in scoping, prefeasibility and feasibility studies for both underground and open pit projects in Canada, Chile, Argentina and Peru.

5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

6. I visited the Baptiste Nickel Project site last June 27, 2023 for a visit duration of 1 day.

7. I am responsible for sections 1.11, 1.12, 1.22.2, 15, 16, 18.2.2, 21.2.3.2, 21.3.3, 25.7, 25.14.1.5, 25.14.2.4, 26.2.2, and 27 of the Technical Report.

8. I am independent of FPX Nickel Corp. as independence is defined in Section 1.5 of NI 43-101.

9. I have had no previous involvement with the Baptiste Nickel Project.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 18, 2023.

"Signed and sealed"

Cristian H. Garcia Jimenez, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Duke Barrett Reimer, P. Eng.

I, Duke Reimer, P.Eng., certify that I am employed as a Senior Engineer with Knight Piésold Ltd. (KP), with an office address of 1400 – 750 West Pender Street, Vancouver, British Columbia, Canada, V6C 2T8.

1. This certificate applies to the technical report titled "Baptiste Nickel Project NI 43-101 Technical Report and Prefeasibility Study" that has an effective date of September 06, 2023 and a report date of October 18, 2023 (the "Technical Report").
2. I graduated from the University of British Columbia in Kelowna, British Columbia, in 2011 with a Bachelor of Applied Science in Civil Engineering.
3. I am a registered Professional Engineer with Engineers and Geoscientists British Columbia (Registration No. 43270).
4. I have practiced my profession for 12 years since graduation. I have been directly involved in performing and overseeing geotechnical engineering design, tailings and water management studies, environmental assessments and monitoring construction activities for mining projects during this time.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I visited the Baptiste Nickel Project site on June 27, 2023.
7. I am responsible for 1.14.4, 1.22.5, 18.7, 18.8, 18.11.1, 21.2.5, 25.14.1.6.1, 25.14.2.5, 26.2.5, and 27 of the Technical Report.
8. I am independent of FPX Nickel Corp. as independence is defined in Section 1.5 of NI 43-101.
9. I have been involved with the Baptiste Nickel Project since 2021, as a consultant, performing and overseeing work related to geotechnical site investigations, tailings and water management studies and the design of project components related to the sections of this Technical Report that I am responsible for preparing.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 18, 2023.

"Signed and sealed"

Duke Barrett Reimer, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Richard FJ Flynn, P. Geo.

I, Richard FJ Flynn, P.Geo., certify that I am employed as a Principal at Next Mine Consulting Ltd., with an office address of 592 17th Ave. W. Vancouver, British Columbia, V5Z 1T5.

1. This certificate applies to the technical report titled "Baptiste Nickel Project NI 43-101 Technical Report and Prefeasibility Study" that has an effective date of September 6, 2023 and a report date of October 18, 2023 (the "Technical Report").

2. I graduated from Saint Mary's University, Halifax, NS, Canada with a B.Sc. degree in 2000 and MBA in 2013; I have a citation in Geostatistics from the University of Alberta in 2018.

3. I am a Professional Geoscientist (P.Geo.) of Engineers & Geoscientists British Columbia; Registration No. 43024.

4. I have practiced my profession for 20 years. I have been directly involved in building geological models and resource estimation for the duration of my career which includes experience in precious metals, base metals, iron ore and metallurgical coal.

5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

6. I visited the Baptiste Nickel Project site between 4th and 7th July, 2022, for a visit duration of four days.

7. I am responsible for sections 1.10, 14, 25.6, 25.14.1.4, and 27 of the Technical Report.

8. I am independent of FPX Nickel Corp. as independence is defined in Section 1.5 of NI 43-101.

9. I have had no previous involvement with the Baptiste Nickel Project.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 18, 2023.

"Signed and sealed"

Richard FJ Flynn, P. Geo.

CERTIFICATE OF QUALIFIED PERSON
David John Baldwin, P. Eng.

I, David John Baldwin, P.Eng., certify that I am employed as a Director of Engineering with Carisbrooke Consulting Inc. ("Carisbrooke"), with an office address of 4339 Morgan Crescent, West Vancouver, British Columbia, Canada.

1. This certificate applies to the technical report titled "Baptiste Nickel Project NI 43-101 Technical Report and Prefeasibility Study" that has an effective date of September 6, 2023 and a report date of October 18, 2023 (the "Technical Report").

2. I graduated from the University of Victoria in 2006 with a Bachelor of Engineering and Queen's University in 2010 with a master's in business administration.

3. I am a Professional Engineer of Engineers and Geoscientist of British Columbia; Registration No. 35139.

4. I have practiced my profession for 17 years. I have been directly involved in over twenty transmission line projects in British Columbia.

5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.

6. I have not visited the Baptiste Nickel Project site.

7. I am responsible for 1.14.3, 18.9.1, 21.2.7.3, and 27 of the Technical Report.

8. I am independent of FPX Nickel Corp. as independence is defined in Section 1.5 of NI 43-101.

9. I have been involved with the Baptiste Nickel Project in a consulting capacity since October 2021 and developed a transmission line route option analysis study.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 18, 2023.

"Signed and sealed"

David John Baldwin, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Paul Mysak, P.Eng.

I, Paul Mysak, P.Eng., certify that I am employed as a Supervising Engineer with Onsite Engineering Ltd., with an office address of 3661 15th Avenue, Prince George, British Columbia, Canada, V2N 1A3.

1. This certificate applies to the technical report titled "Baptiste Nickel Project NI 43-101 Technical Report and Prefeasibility Study" that has an effective date of September 6, 2023 and a report date of October 18, 2023 (the "Technical Report").
2. I graduated from the University of New Brunswick, Fredericton, New Brunswick, in 2007 with a Bachelor of Science in Forest Engineering.
3. I am a Professional Engineer of Engineers and Geoscientists British Columbia (Registration No. 38728).
4. I have practiced my profession for 14 years since graduation. I have been directly involved in the field location, survey, design, and construction of resource roads and bridges. Recent relevant experience includes acting as Engineer of Record (EOR) for the general arrangement design and construction of the 235m (800') multispan bridge crossing the Stuart River for SA Energy Group near Vanderfhoof, BC, and EOR for the design and construction of a 1.5km double lane bypass road for Conuma Resources near Tumbler Ridge, BC. From 2020 to 2021, I worked for Hampton Lumber on the prescription design and construction field reviews for the recently completed upgrade to the Austin FSR which is used to access the Nickel Project site.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I visited the Baptiste Nickel Project site between November 25th and 26th, 2021 for a visit duration of 2 days.
7. I am responsible for sections 1.14.1, 18.2.1, 21.2.7.1, and 27 of the Technical Report related to the off-site access roads.
8. I am independent of FPX Nickel Corp. as independence is defined in Section 1.5 of NI 43-101.
9. I have been involved with the Baptiste Nickel Project from November 2021 to March 2022 when, as a consultant, one of the projects I worked on during that time frame was the Access Road Summary Report Revision 1 Issued on March 16, 2022 by Onsite Engineering Ltd. for FPX Nickel.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 18, 2023.

"Signed and sealed"

Paul Mysak, P.Eng.

Onsite Engineering Ltd. Permit to Practice No. 1002678, Engineers and Geoscientists British Columbia

CERTIFICATE OF QUALIFIED PERSON
Harold Rolf Schmitt, P. Geo.

I, Harold Rolf Schmitt, P.Geo., certify that I am employed as a Technical Director within ERM Consultants Canada Ltd., with an office address of # 1000 – 1100 Melville Street, Vancouver, British Columbia, Canada, V6E 4A6.

1. This certificate applies to the technical report titled "Baptiste Nickel Project NI 43-101 Technical Report and Prefeasibility Study" that has an effective date of September 6, 2023 and a report date of October 18, 2023 (the "Technical Report").
2. I graduated from the Univ. of British Columbia – Bachelor of Science (B.Sc.) Geology (1977), and a Master of Science (M.Sc.) Regional Planning (1985), and Univ. of Ottawa - Master of Science (M.Sc.) Exploration Geochemistry (1993).
3. I am a Professional Geoscientist registered with the Association of Professional Engineers and Geoscientists of British Columbia, Registration no.19824 and have been a registrant in good standing continuously since 17/11/1992.
4. I have practiced my profession for 46 years since graduation. My relevant experience includes:
   - Exploration Geologist, Texasgulf Inc. and Kidd Creek Mines, 1975-1981: geology and baseline surveys
   - Geochemist, Geological Survey of Canada, 1985-1989: baseline surveys, MLARD
   - Sr Land-Use Geologist, BC Ministry of Energy and Mines, 1981-1984; 1989-2005: mine regulation, land use
   - Technical Director, Geoscience/ Permitting, Rescan / ERM …2005-2023: EA/Mine and infrastructure permitting.
   - NI 43-101 Section 20 QP on 13 PEA, PFS and FS technical reports on multiple BC, NWT, ONT and international mine projects during recent 10 years.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the Baptiste Nickel Project site.
7. I am responsible for sections: 1.16, 1.22.6, 3.2, 20, 24.2.10, 25.9, 25.14.1.8, 25.14.2.6, 26.2.6, and 27 of the Technical Report.
8. I am independent of FPX Nickel Corp. as independence is defined in Section 1.5 of NI 43-101.
9. I have been involved with the Baptiste Nickel Project since 2021, as an independent consultant to FPX on an intermittent basis. I co-authored an internal regulatory and permitting strategy and environmental gap analysis report for FPX (2021-2022) and updated regulatory schedule in 2023.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 18, 2023

"Signed and sealed"

Rolf Schmitt, P.Geo.

ERM Permit to Practice No. 1001271, Engineers and Geoscientists British Columbia

Important Notice

This report was prepared as National Instrument 43-101 Technical Report for FPX Nickel Corp. (FPX) by Ausenco Engineering Canada Inc. (Ausenco), International Metallurgical & Environmental Inc. (IMI), TechSer Mining Consultants Ltd. (TechSer), Knight Piésold Ltd. (KP), Equity Exploration Consultants Ltd. (Equity Exploration), Next Mine Consulting Ltd. (Next Mine), Carisbrooke Consulting Inc. (Carisbrooke), Onsite Engineering Ltd. (Onsite Engineering), and ERM Consultants Canada Ltd. (ERM), collectively the Report Authors. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by FPX subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party are at that party's sole risk.

Table of Contents

1 SUMMARY ... 20
1.1 Introduction ... 20
1.2 Property Description, Location and Ownership ... 21
1.3 History ... 23
1.4 Geology and Mineralization ... 24
1.5 Exploration ... 25
1.6 Drilling ... 25
1.7 Sample Preparation, Analysis and Security ... 26
1.8 Data Verification ... 27
1.9 Mineral Processing and Metallurgical Testwork ... 28
1.10 Mineral Resource Estimate ... 30
1.11 Mineral Reserve Statement ... 32
1.12 Mining Methods ... 33
1.13 Recovery Methods ... 35
1.14 Project Infrastructure ... 37
1.14.1 Site Access ... 37
1.14.2 Site Infrastructure ... 37
1.14.3 Site Power ... 39
1.14.4 Tailings Facility ... 39
1.14.5 Freshwater Supply System ... 39
1.15 Market Studies and Contracts ... 40
1.16 Environmental Studies, Permitting, and Engagement ... 42
1.16.1 Environmental Baseline Studies ... 42
1.16.2 Environmental Assessment and Permitting ... 42
1.16.3 Indigenous Engagement ... 44
1.16.4 Mine Design and Closure Planning ... 44
1.17 Capital and Operating Costs ... 45
1.17.1 Capital Cost Estimate ... 45
1.17.2 Operating Cost Estimate ... 46
1.18 Economic Analysis ... 46
1.19 Adjacent Properties ... 50
1.20 Refinery Option ... 51
1.21 Conclusions and Interpretations ... 54
1.22 Recommendations ... 54
1.22.1 Geology and Resource ... 54

1.22.2 Mining ... 55
1.22.3 Metallurgy ... 55
1.22.4 Process and Infrastructure ... 56
1.22.5 Waste and Water Management ... 57
1.22.6 Environment and Engagement ... 57
1.22.7 Refinery Option ... 58

2 INTRODUCTION ... 59
2.1 Introduction ... 59
2.2 Terms of Reference ... 59
2.3 Qualified Persons ... 59
2.4 Site Visits and Scope of Personal Inspection ... 61
2.5 Effective Dates ... 61
2.6 Information Sources and References ... 62
2.7 Previous Technical Reports ... 62
2.8 Currency, Units, Abbreviations and Definitions ... 62

3 RELIANCE ON OTHER EXPERTS ... 70
3.1 Introduction ... 70
3.2 Environmental, Permitting, Closure, and Social and Community Impacts ... 70
3.3 Taxation ... 71
3.4 Marketing ... 71

4 PROPERTY DESCRIPTION AND LOCATION ... 72

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY ... 75
5.1 Physiography ... 75
5.2 Accessibility ... 75
5.3 Climate ... 76
5.4 Local Resources and Infrastructure ... 77

6 HISTORY ... 79
6.1 Early Chromium, PGE, and Gold Exploration ... 79
6.2 Awaruite Exploration ... 80
6.3 Historical Resource Estimates ... 81
6.4 Production ... 81

7 GEOLOGICAL SETTING AND MINERALIZATION ... 82
7.1 Regional Geology ... 82
7.2 Regional Metallogeny ... 83
7.3 Property Geology ... 83
7.3.1 Lithology ... 83
7.3.2 Structural Geology ... 87

7.3.3 Alteration ... 87
7.4 Property Mineralization ... 88
8 DEPOSIT TYPES ... 90
8.1 Deposit Model ... 90
9 EXPLORATION ... 92
9.1 Surface Geological and Geochemical Surveys ... 92
9.2 Airborne Surveys ... 95
9.2.1 Gradient magnetics ... 95
9.2.2 LiDAR ... 96
9.3 Ground-Based Geophysical Surveys ... 97
9.3.1 Ground-based Induced Polarization (IP) and Resistivity ... 97
9.3.2 Ground-based Magnetic Survey ... 98
9.4 Downhole Surveys ... 99
9.4.1 Physical Rock Properties ... 99
9.4.2 Induced Polarization (IP) and Resistivity ... 100
9.4.3 Magnetics ... 102
9.4.4 Televiewer Surveys ... 103
10 DRILLING ... 104
10.1 Baptiste Deposit ... 104
10.1.3 2010 Drilling ... 105
10.1.4 2011 Drilling ... 107
10.1.5 2012 Drilling ... 109
10.1.6 2017 Drilling ... 111
10.1.7 2021 Drilling ... 112
10.1.8 2022 Drilling ... 113
10.2 Van Target ... 113
10.2.1 2021 Drilling ... 113
10.2.2 2022 Drilling ... 116
10.3 B Target ... 117
10.4 Sid Target ... 117
11 SAMPLE PREPARATION, ANALYSES AND SECURITY ... 119
11.1 Sampling Methods ... 119
11.2 Sample Shipping and Security ... 120
11.3 Analytical Methods ... 120
11.4 Quality Assurance and Quality Control (QA/QC) ... 121
11.4.1 Certified Reference Materials ... 122
11.4.2 Non-certified Standards ... 124
11.4.3 Replicates ... 124

11.4.4 Blanks ... 125
11.4.5 Duplicates ... 128
11.4.6 Magnetic separates ... 130
11.4.7 Specific gravity ... 131
11.5 Data Adequacy ... 131
12 DATA VERIFICATION ... 132
12.1 Legacy Data ... 132
12.2 Verification performed by the QP ... 136
13 MINERAL PROCESSING AND METALLURGICAL TESTING ... 137
13.1 Introduction ... 137
13.1.1 Metallurgical Development History ... 138
13.1.2 Metallurgical Accounting ... 139
13.2 Summary of Metallurgical Test Results ... 140
13.3 Sample Selection ... 142
13.3.1 2021 Life-of-Mine Composite ... 143
13.3.2 Bulk Outcrop ... 146
13.3.3 Mine Phase Composites ... 147
13.4 Mineralogy ... 150
13.4.1 Mineral Content ... 150
13.4.2 Elemental Analysis of Nickel Minerals ... 153
13.4.3 Mineral Grain Size ... 154
13.4.4 Mineral Liberation ... 156
13.5 Comminution ... 159
13.5.1 JKDW / SMC Test ... 160
13.5.2 Bond Ball Mill Grindability Test ... 160
13.5.3 Bond Rod Mill Grindability Test ... 161
13.5.4 MacPherson Autogenous Grindability Test ... 161
13.5.5 Bond Low-Energy Impact Crusher Test ... 161
13.5.6 Bond Abrasion Test ... 162
13.5.7 Rock Quality Designation ... 162
13.6 Magnetic Separation ... 162
13.6.1 Introduction ... 162
13.6.2 Primary Magnetic Separation ... 163
13.6.3 Cleaner Magnetic Separation ... 172
13.7 Gravity Separation ... 177
13.8 Flotation ... 178
13.8.1 PEA Bench Testing ... 178
13.8.2 PFS Bench Testing ... 179

13.9 Flotation Tailings Leach 185
13.10 Product Quality 187
13.10.1 Awaruite Concentrate 187
13.10.2 Mixed Hydroxide Precipitate 188
13.11 Recovery Model 188
14 MINERAL RESOURCE ESTIMATES 191
14.1 Introduction 191
14.2 Key Assumptions and Basis of Estimate 191
14.3 Geological Model 191
14.4 Exploratory Data Analysis 196
14.5 Density Assignment 199
14.6 Grade Capping and Outlier Restrictions 199
14.7 Variography 200
14.8 Estimation/Interpolation Methods 201
14.8.1 Estimation Methodology 202
14.9 Block Model Validation 208
14.10 Classification of Mineral Resources 211
14.11 Reasonable Prospects for Eventual Economic Extraction 212
14.12 Mineral Resource Statement 212
14.13 Comparison with the Previous Mineral Resource Estimate 213
14.14 Comments on the Mineral Resource Estimate 215
15 MINERAL RESERVE ESTIMATES 216
15.1 Geotechnical Considerations 216
15.2 Dilution and Mining Loss 216
15.3 Net Smelter Return (NSR) 216
15.3.1 Economic Assumptions 216
15.3.2 Metallurgical Recoveries 217
15.4 Cut-Off Grade 217
15.5 Pit Optimization 217
15.6 Ultimate Pit Design 219
15.7 Mineral Reserve Statement 220
15.8 Factors that May Affect the Mineral Reserves Estimate 221
16 MINING METHODS 222
16.1 Key Design Criteria 222
16.1.1 Net Smelter Revenues (NSR) Formula 223
16.1.2 Dilution and Mining Losses 223
16.1.3 Geotechnical Considerations 223
16.2 Pit Optimization 224

16.2.1 Pit Limits ... 225
16.2.2 Sensitivities ... 230
16.3 Mine Design ... 231
16.3.1 Mine Phase 1 Design ... 233
16.3.2 Mine Phase 2 Design ... 234
16.3.3 Mine Phase 3 Design ... 235
16.3.4 Mine Phase 4 Design ... 236
16.3.5 Mine Phase 5 Design ... 237
16.3.6 Mine Phase 6 Design ... 238
16.3.7 Mine Phase 7 Design ... 239
16.3.8 Mine Phase 8 Design ... 240
16.3.9 Pit Summary ... 241
16.4 Mine Waste Management ... 243
16.4.1 Operational Ore Stockpile ... 243
16.4.2 Tailings Facility Dams and Internal Causeway ... 243
16.4.3 Overburden Stockpile ... 243
16.4.4 In-Pit Waste Backfill ... 244
16.5 Production Schedule ... 244
16.5.1 Pre-Production ... 244
16.5.2 Production ... 245
16.5.3 End-of-Period Maps ... 247
16.6 Mining Operations ... 256
16.6.1 Drilling ... 256
16.6.2 Blasting ... 257
16.6.3 Loading ... 257
16.6.4 Hauling ... 258
16.6.5 Auxiliary Equipment ... 261
16.7 Mining Personnel ... 262
17 RECOVERY METHODS ... 264
17.1 Overview ... 264
17.2 Plant Design ... 265
17.3 Process Flowsheet ... 267
17.3.1 Primary Crushing and Coarse Ore Stockpile ... 270
17.3.2 Primary Grinding ... 271
17.3.3 Magnetic Separation and Regrind ... 273
17.3.4 Awaruite Flotation ... 277
17.3.5 Awaruite Concentrate Dewatering and Loadout ... 279
17.3.6 Flotation Tailings Leaching ... 280

17.3.7 Solution Purification and MHP Precipitation ... 282
17.3.8 Reagents and Grinding Media ... 283
17.3.9 Services and Utilities ... 286

18 PROJECT INFRASTRUCTURE ... 290

18.1 Introduction ... 290
18.2 Site Access ... 292
18.2.1 Off-site Access Roads ... 292
18.2.2 Mine Roads ... 293
18.2.3 On-site Plant Roads ... 294
18.2.4 Railway ... 294
18.2.5 Airports ... 294
18.3 Mine Maintenance Facilities ... 294
18.4 Process Infrastructure ... 295
18.5 Site Infrastructure ... 296
18.5.1 Accommodations ... 296
18.5.2 Sewage and Waste Management ... 297
18.6 Stockpiles ... 297
18.7 Tailings Facility ... 297
18.7.1 Site Selection ... 298
18.7.2 Facility Design ... 298
18.7.3 Tailings Distribution ... 301
18.8 Waste Rock Storage Facilities ... 301
18.9 Power and Electrical ... 302
18.9.1 Off-site Power and Distribution ... 302
18.9.2 On-site Power and Distribution ... 302
18.10 Fuel ... 302
18.11 Water Supply and Management Systems ... 302
18.11.1 Water Management Systems ... 302
18.11.2 Open Pit Dewatering System ... 303
18.11.3 Freshwater Supply System ... 303
18.12 Hazard Considerations ... 304
18.13 Concentrate Bagging, Outloading, and Logistics ... 304

19 MARKET STUDIES AND CONTRACTS ... 306

19.1 Introduction ... 306
19.1.1 Baptiste Awaruite Concentrate Product Overview ... 306
19.1.2 Baptiste MHP Product Overview ... 306
19.1.3 Other Potential Products ... 307
19.2 Characteristics of the Awaruite Product ... 307

19.3 Global Nickel Supply and Demand 309
19.4 LME Nickel Price for PFS Economic Analysis 309
19.5 Baptiste Product Payability Analysis 309
19.5.1 Baptiste Awaruite Concentrate Product 310
19.5.2 Baptiste MHP Product Payability Analysis 311
19.6 Selling Prices Used for PFS Economic Analysis 311
19.7 Contracts 312
19.8 Qualified Person Comments 312

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT 313

20.1 Environmental Baseline Studies 313
20.1.1 2011 to 2014 Baseline Studies 313
20.1.2 2021 to Present Baseline Studies 314
20.1.3 Outstanding Areas of Study Required to Fulfill the Environmental Assessment Requirements 318
20.2 Environmental Assessment and Permitting 319
20.2.1 Environmental Assessment Process 319
20.2.2 Mine Permitting 321
20.3 Indigenous Engagement 323
20.4 Mine Design and Closure Planning 324
20.4.1 Mine Waste and Water Management 324
20.4.2 Closure and Reclamation Plan 324
20.5 Next Steps 325
20.5.1 Cultural and Environmental Baseline Studies – Next Steps 325

21 CAPITAL AND OPERATING COSTS 327

21.1 Introduction 327
21.2 Capital Costs 327
21.2.1 Capital Cost Estimate Input 327
21.2.2 Capital Cost Estimate Summary 328
21.2.3 Area 1000 – Direct Costs, Mining 329
21.2.4 Area 2000 – Direct Costs, Process Plant 330
21.2.5 Area 3000 – Direct Costs, Tailings Facility 332
21.2.6 Area 4000 – Direct Costs, On-site Infrastructure 333
21.2.7 Area 5000 – Direct Costs, Off-site Infrastructure 335
21.2.8 Area 7000 to 9000 – Indirect Costs 336
21.2.9 Exclusions 341
21.3 Operating Costs 341
21.3.1 Operating Cost Estimate Summary 341
21.3.2 Basis of Estimate 343
21.3.3 Mine Operating Costs 343

21.3.4 Process Operating Costs 344
21.3.5 General and Administrative Operating Costs 346
21.3.6 Concentrate Handling and Transport 346
21.3.7 Owner's Costs 347
21.3.8 Closure and Reclamation Planning 348
22 ECONOMIC ANALYSIS 350
22.1 Forward-Looking Information Cautionary Statements 350
22.2 Methodologies Used 351
22.3 Financial Model Parameters 351
22.3.1 Taxes 351
22.3.2 Working Capital 352
22.3.3 Royalty 352
22.4 Economic Analysis 352
22.5 Sensitivity Analysis 357
23 ADJACENT PROPERTIES 361
23.1 Brynelsen and Bell Claims 361
23.2 MAC Property 361
23.3 Grid Battery Claims 362
24 OTHER RELEVANT DATA AND INFORMATION 363
24.1 Project Execution Plan 363
24.1.1 Execution Strategy 363
24.1.2 Project Risk Identification 365
24.1.3 Project Team 365
24.1.4 Project Alignment 365
24.1.5 Sponsor Group 365
24.1.6 Governance 366
24.1.7 External Relations 366
24.1.8 Construction Execution Strategy 366
24.2 Refinery Option 368
24.2.1 Introduction 368
24.2.2 Property Description and Location 369
24.2.3 Mineral Processing and Metallurgical Testing 369
24.2.4 Mineral Resource Estimates 381
24.2.5 Mineral Reserve Estimates 381
24.2.6 Mining Methods 381
24.2.7 Recovery Methods 381
24.2.8 Project Infrastructure 392
24.2.9 Market Studies and Contracts 394

24.2.10 Environmental Studies, Permitting, and Social and Community Impact 396
24.2.11 Capital and Operating Costs 397
24.2.12 Economic Analysis 405

25 INTERPRETATION AND CONCLUSIONS 411

25.1 Introduction 411
25.2 Mineral Tenure, Surface Rights, Water Rights, Royalties 411
25.3 Geology and Mineralization 411
25.4 Exploration, Drilling, and Analytical Data Collection in Support of Resource Estimation 411
25.5 Metallurgical Testwork 412
25.6 Mineral Resource Estimate 412
25.7 Mining 412
25.7.1 Geotechnical Considerations 412
25.7.2 Mine Plan 413
25.8 Recovery Methods 413
25.9 Environmental, Permitting and Social Considerations 414
25.10 Capital Cost Estimate 414
25.11 Operating Cost Estimate 414
25.12 Economic Analysis 414
25.13 Refinery Option 415
25.14 Risk and Opportunities 415
25.14.1 Risks 415
25.14.2 Opportunities 418

26 RECOMMENDATIONS 422

26.1 Path Forward – Budget and Schedule 422
26.2 Path Forward – Scope of Work 422
26.2.1 Geology and Resource 422
26.2.2 Mining 423
26.2.3 Metallurgy 423
26.2.4 Process and Infrastructure 424
26.2.5 Mine Waste and Water Management 425
26.2.6 Environment and Engagement 426
26.2.7 General 426
26.3 Refinery Option 426

27 REFERENCES 428

List of Tables

Table 1-1: Summary of Drilling Completed by FPX on the Decar Property 26
Table 1-2: Overview of Decar Drill Core Sampling Campaigns 26
Table 1-3: Metallurgical Results for Life-of-Mine Composite Sample 30
Table 1-4: 2022 Baptiste Deposit Pit-Constrained Mineral Resource Estimate 31
Table 1-5: Indicated and Inferred Resources for the Baptiste Deposit 32
Table 1-6: Baptiste Nickel Project Reserve Estimate 33
Table 1-7: Summary of Mine Phase Tonnages and Grades 34
Table 1-8: Projected Analyst LME Benchmark Price 40
Table 1-9: Historical Premium / Discount for Standard FeNi 41
Table 1-10: Selling Price for Nickel Products in PFS Economic Analysis 41
Table 1-11: Summary of Capital Costs 45
Table 1-12: Operating Cost Summary – Phase 1 Average 46
Table 1-13: Operating Cost Summary – Phase 2 Average 46
Table 1-14: Operating Cost Summary – LOM Average 46
Table 1-15: Summary of Project LOM Cashflow Assumptions and Results 47
Table 1-16: Post-Tax Sensitivity Analysis Results 49
Table 1-17: Capital Cost Estimate Summary – Refinery Only 52
Table 1-18: Operating Cost Estimate Summary – Refinery Only 52
Table 1-19: Summary of Project LOM Cashflow Assumptions and Results – Refinery Option 53
Table 1-20: Proposed Feasibility Study Budget 54
Table 2-1: Report Contributors 60
Table 2-2: Site Visits 61
Table 2-3: Summary of Previous Technical Reports 62
Table 2-4: List of Abbreviations 63
Table 6-1: Historical Resource Estimates for the Baptiste Deposit 81
Table 9-1: Overview of FPX Surface Samples from the Decar Property 92
Table 9-2: Overview of FPX Airborne Surveys 95
Table 9-3: Overview of FPX Ground-Based Geophysical Surveys 97
Table 9-4: Overview of FPX Downhole Geophysical Surveys 99
Table 9-5: Overview of FPXs Downhole Magnetic Susceptibility Surveys, with Intensities Measured as SI Units .. 103
Table 10-1: Summary of Drilling Completed by FPX on the Decar Property 104
Table 10-2: Composites >20 DTR Ni%*m from 2010 Drilling on the Decar Property 107
Table 10-3: Composites >20 DTR Ni%*m from 2011 Drilling on the Decar Property 108
Table 10-4: Composites >20 DTR Ni%*m from 2012 Drilling on the Decar Property 110
Table 10-5: Composites >20 DTR Ni%*m from 2017 Drilling on the Decar Property 111
Table 10-6: Composites >20 DTR Ni%*m from 2021 Drilling on the Baptiste Deposit 113
Table 10-7: Composites > 20 DTR Ni%*m from 2021 Drilling on the Van Target 116
Table 10-8: Composites >20 DTR Ni%*m from 2022 Drilling on the Van Target 117
Table 10-9: Composites >20 DTR Ni%*m from 2010 Drilling on the Sid target 118

Table 11-1: Overview of Decar Drill Core Sampling Campaigns ... 119
Table 11-2: Overview of Analytical Methods Used by FPX for Drill Core ... 121
Table 11-3: Overview of CRM Performance (N >25) for the Decar Property ... 122
Table 11-4: Overview of NCS Performance (N > 25) for Decar Property Analyses ... 124
Table 11-5: Overview of Replicate Performance (N > 25) For Decar Property Analyses ... 125
Table 11-6: Overview of Blank Performance for Select Decar Property Analyses ... 126
Table 11-7: Summary Statistics for Duplicates Samples from the Decar Property ... 129
Table 11-8: Summary Statistics for Magnetic Separate Duplicates from the Decar Property ... 131
Table 12-1: Field Duplicate Assays Taken During Site Visit July 2022 ... 134
Table 13-1: Metallurgical Testwork Summary Table ... 138
Table 13-2: Metallurgical Results for Life-of-Mine Composite Sample ... 142
Table 13-3: List of Samples Used In Metallurgical Testwork Campaigns ... 143
Table 13-4: Distribution of Nickel by Mineral in Metallurgical Samples ... 152
Table 13-5: Average Elemental Grades of Nickel Minerals from the Mine Phase Composite Samples ... 153
Table 13-6: Summary of RQD Statistics by Mine Phase ... 162
Table 13-7: Recleaner Magnetic Concentrates Produced in Variability Testing ... 177
Table 13-8: Composition of PFS Bench Testing Flotation Feed ... 179
Table 13-9: Typical Flotation Conditions Used In PFS Bench Testing (Test F35) ... 180
Table 13-10: Typical Open Circuit Flotation Test Results (Test F35) ... 180
Table 13-11: Locked Cycle Flotation Test Results ... 181
Table 13-12: Flotation Tailings Composition ... 184
Table 13-13: Comparison of Standard and Alternative Flotation Regime Locked Cycle Flotation Results ... 184
Table 13-14: Selective Nickel Assay on Flotation Tailings ... 185
Table 13-15: Awaruite Concentrate Product Specifications ... 187
Table 13-16: Life-of-Mine Ore Grade Mass Balance ... 189
Table 14-1: Summary of Drillhole and Sample Totals Used in 2022 Resource Estimate ... 191
Table 14-2: Modelled Domains Drill Length vs Drillhole Intervals ... 195
Table 14-3: Length Weighted Assays in Estimated Domains ... 196
Table 14-4: Descriptive Statistics of 5 m Composites from Mineralized Domains ... 197
Table 14-5: Density Assignments for Each Lithological Domain ... 199
Table 14-6: Decile Analysis (0-100) for Mineralized Domains ... 200
Table 14-7: Centile Analysis (90-100) for Mineralized Domains ... 200
Table 14-8: Block Model Extents ... 202
Table 14-9: Domain Wireframe Volume vs Block Model Volume ... 202
Table 14-10: Grade Model Search Parameters ... 206
Table 14-11: 2022 Baptiste Deposit Pit-Constrained Mineral Resource Estimate* ... 212
Table 14-12: Indicated and Inferred Resources for the Baptiste Deposit ... 213
Table 14-13: Comparison of 2018 vs 2022 Indicated and Inferred Resources ... 213
Table 15-1: Key Economic Assumptions for NSR Calculations ... 217
Table 15-2: Baptiste Nickel Project Reserve Estimate ... 221
Table 16-1: Mine Design Criteria for PFS Optimization ... 223

Table 16-2:	Mine Design Pit Slope Recommendations	224
Table 16-3:	Whittle Optimization Parameters	224
Table 16-4:	Pit Optimization Run Results	226
Table 16-5:	Pit Optimization Sensitivity Criteria	230
Table 16-6:	Key Mine Design Criteria for PFS Mine Design	231
Table 16-7:	High level Mine Design Scenarios for Baptiste PFS	232
Table 16-8:	Summary of Mine Phase Tonnages and Grades	241
Table 16-9:	Baptiste Ore Tonnage and Grade at Varying NSR Cut-off Grades	242
Table 16-10:	Storage Capacity by Waste and Ore Storage Facility	244
Table 16-11:	Summary Table of Mine Production per Annum of Operation	246
Table 16-12:	Peak Primary Production Equipment Counts	256
Table 16-13:	Drilling Parameters	257
Table 16-14:	Truck Haulage Design Criteria	258
Table 16-15:	Haulage Design Criteria	260
Table 16-16:	Truck Shovel Match	260
Table 16-17:	Average Minnig Personnel by Project Phase	263
Table 17-1:	Summary of Major Process Design Criteria	265
Table 17-2:	Reagents Consumption	284
Table 17-3:	Power Requirements and Energy Consumption	288
Table 18-1:	Mine Maintenance Facilities	294
Table 18-2:	Process Infrastructure Buildings	296
Table 18-3:	General Facilities	296
Table 18-4:	Tailings Facility Storage Requirements (KP, 2023)	300
Table 19-1:	Production Specification for the Awaruite Concentrate Briquettes	308
Table 19-2:	Nickel Content Comparison of Various Nickel Products	308
Table 19-3:	Projected Analyst LME Benchmark Price	309
Table 19-4:	Historical Premium / Discount for Standard FeNi	311
Table 19-5:	Selling Price for Nickel Products in PFS Economic Analysis	312
Table 20-1:	Summary of 2011-2014 Environmental Baseline Studies	313
Table 20-2:	Summary of 2021-2023 Environmental Baseline Studies	315
Table 20-3:	Possible Provincial Authorizations	322
Table 20-4:	Possible Federal Authorizations	323
Table 21-1:	Estimate Exchange Rates	328
Table 21-2:	Summary of Total Capital Costs	329
Table 21-3:	Mining Capital Cost Estimate	329
Table 21-4:	Process Capital Cost Estimate	332
Table 21-5:	Tailings Facility Capital Cost Estimate	332
Table 21-6:	On-site Infrastructure Capital Cost Estimate	333
Table 21-7:	Off-site Infrastructure Capital Cost Estimate	335
Table 21-8:	Indirect Costs	336
Table 21-9:	Indirect Costs	339

Table 21-10: Owner's Costs ... 339
Table 21-11: Contingency by WBS ... 340
Table 21-12: Total Growth Allowances ... 341
Table 21-13: Life-Of-Mine Operating Cost and AISC ... 342
Table 21-14: Operating Cost Summary – Phase 1 Average ... 342
Table 21-15: Operating Cost Summary – Phase 2 Average ... 342
Table 21-16: Operating Cost Summary – LOM Average ... 342
Table 21-17: Mining Operating Costs ... 344
Table 21-18: Processing Labour by Department ... 344
Table 21-19: Processing Labour Operating Cost ... 345
Table 21-20: Consumables and Reagents ... 345
Table 21-21: Maintenance Consumable Costs ... 345
Table 21-22: G&A Operating Costs ... 346
Table 21-23: Owner's Cost ... 347
Table 22-1: Summary of Project LOM Cashflow Assumptions and Results ... 354
Table 22-2: Projected LOM Post-Tax Unlevered Free Cash Flow ... 355
Table 22-3: Projected LOM Post-Tax Unlevered Free Cash Flow (Continued) ... 356
Table 22-4: Pre-Tax Sensitivity Analysis Results ... 357
Table 22-5: Post-Tax Sensitivity Analysis Results ... 359
Table 24-1: Refining Metallurgical Testwork Summary ... 370
Table 24-2: List of Samples Used in Refining Metallurgical Testwork Campaigns ... 371
Table 24-3: Inferred mineralogy of Awaruite Concentrate ... 373
Table 24-4: Pressure Oxidation Leach Results Summary (2023 Program) ... 373
Table 24-5: Pressure Oxidation Leach Results Summary (2020 Program) ... 374
Table 24-6: Atmospheric Oxidation Leach Results Summary ... 374
Table 24-7: Copper Cementation Results Summary ... 375
Table 24-8: Aluminum and Chromium Removal Results Summary – Part 1 Leaching ... 376
Table 24-9: Aluminum and Chromium Removal Results Summary – Part 2 Precipitation ... 376
Table 24-10: PLS Solution Composition ... 377
Table 24-11: Nickel Sulphate Crystallization Results Summary ... 379
Table 24-12: Nickel Scavenging Results Summary ... 379
Table 24-13: Summary of Major Processing Design Criteria – Refinery Option ... 382
Table 24-14: Building Footprint by Area – Refinery Option ... 393
Table 24-15: Baptiste Nickel Sulphate Crystal Quality vs. Typical Target Specification ... 395
Table 24-16: Capital Cost Summary – Refinery Option ... 398
Table 24-17: Capital Cost Summary, by WBS Code – Refinery Option ... 399
Table 24-18: Operating Cost Summary (Acid Credits Not Included) ... 402
Table 24-19: Power Cost Breakdown ... 402
Table 24-20: Reagents and Consumables Cost Breakdown ... 403
Table 24-21: Labour Cost Breakdown – Total Workforce ... 403
Table 24-22: Maintenance Consumables Cost Breakdown ... 404

Table 24-23: Product Transportation Cost Breakdown 404
Table 24-24: G&A Cost Breakdown 404
Table 24-25: Summary of Project LOM Cashflow Assumptions and Results – Refinery Option 408
Table 24-26: Post-Tax Sensitivity Analysis Results – PFS Base Case + Refinery Option Scenario 409
Table 25-1: Mine Design Pit Slope Recommendations 413
Table 26-1: Proposed Feasibility Study Budget 422

List of Figures

Figure 1-1: Location Map for the Decar Property 22
Figure 1-2: Decar Property Tenure Map 23
Figure 1-3: Baptiste Concentrator Block Flow Diagram 29
Figure 1-4: Overall Process Flow Diagram 35
Figure 1-5: Overall Site Layout 38
Figure 1-6: Projected Post-Tax Undiscounted Cash Flow 48
Figure 1-7: Location of Adjacent Properties 50
Figure 4-1: Location Map for the Decar Property 73
Figure 4-2: Decar Property Tenure Map (A List of Claims is Provided in Appendix A) 74
Figure 5-1: Decar Property Access and Infrastructure Map 76
Figure 5-2: Terrain at the Baptiste Deposit 78
Figure 7-1: Regional Geological Setting of the Decar Property Showing the Cache Creek (CC), Stikine (ST) and Quesnel (QN) Terranes 82
Figure 7-2: Local Bedrock Geology of the Decar Property. 84
Figure 7-3: Property-Scale Bedrock Geology, Structure, and Mineral Deposits of the Decar Property. 85
Figure 9-1: Decar Property Rock Sample Map 93
Figure 9-2: Total Magnetic Intensity (TMI) from the 2010 Airborne Survey 96
Figure 9-3: Plan Map of Contoured Chargeability at N = 5 Separations for the Baptiste and Sidney Grids. 98
Figure 9-4: Example of Physical Rock Property Strip Logs Provided by DGI 100
Figure 9-5: Example of a Downhole IP Strip Log Generated for Drillhole 11BAP003 102
Figure 10-1: Drill Collar Map for the Baptiste Deposit 105
Figure 10-2: Section Through the Baptiste Deposit Showing Multiple Drillholes that all end in Mineralization 106
Figure 10-3: Plan Map for the Van Target Showing Holes Drilled in 2021 and 2022 114
Figure 10-4: Section Through the Van Target Showing Two Drillholes that End in Mineralization 115
Figure 11-1: Shewhart control charts of Z-scores for CRM used to track (clockwise from top left) Nickel by fusion/XRF, whole rock Ni by Fusion/ICP, Iron by Fusion/XRF, and Cobalt by Fusion-XRF 123
Figure 11-2: Control charts of detection limit multiples for blanks used to monitor contamination for (clockwise from top left) nickel by fusion/XRF, whole rock Ni by fusion/ICP, iron by fusion/XRF, and cobalt by Fusion-XRF. 127

Figure 11-3: X-Y Plots Showing Parent and Daughter Ni Assays for Field, Coarse, and Pulp Duplicates by Fusion-XRF as well as Magnetic Separate (“Mag Sep”) Weights for Check Duplicates. 130
Figure 12-1: Images of Core from Hole BAP21087 132
Figure 12-2: a) Reclaimed Drill Pad with Casing Circled, and b) Assays Taken as Field Duplicates for Processing at Act Labs 133
Figure 12-3: Scatterplots of Original vs Field duplicates for DTR Ni%, ICP Ni%, DTR Fe% and DTR Co% 135
Figure 12-4: Photos from Voordouw’s 2018 Site Visit 136
Figure 13-1: Block Flow Diagram of the Baptiste PFS Processing Route 141
Figure 13-2: Histograms Comparing Grades of the 2021 LOM Composite to the PFS Mine Plan 144
Figure 13-3: Histograms Comparing Rock and Alteration Properties of the 2021 LOM Composite to the PFS Mine Plan 145
Figure 13-4: Histograms Comparing Spatial Parameters of the 2021 LOM Composite to the PFS Mine Plan 146
Figure 13-5: 2020 PEA Mine Phase Pit Shells – Plan View 147
Figure 13-6: PFS Mine Phase Pit Shells – Plan View 148
Figure 13-7: PFS Mine Phase Pit Shells – Looking North-East 149
Figure 13-8: Distribution of Intervals Contained in the PFS Mine Phases in Mine Phase Composite Samples 149
Figure 13-9: Mine Phase Schedule 150
Figure 13-10: Mineralogy Content of Metallurgical Samples 151
Figure 13-11: Distribution of Nickel by Mineral in Metallurgical Samples 152
Figure 13-12: Distribution of Iron by Mineral in Metallurgical Samples 153
Figure 13-13: Awaruite Grain Size Distribution (2D Analysis) 154
Figure 13-14: Relationship Between Total Nickel Reporting as DTR Ni and Proportion of Very Fine-Grained Awaruite 155
Figure 13-15: Magnetite Grain Size Distribution (3D analysis) 156
Figure 13-16: Awaruite Liberation by Class of Association 157
Figure 13-17: Magnetite Liberation by Class of Association 158
Figure 13-18: Liberated Awaruite by Sizing Fraction 158
Figure 13-19: Liberated Magnetite by Sizing Fraction 159
Figure 13-20: Relationship Between Primary Grind Size and Total Nickel Recovery 163
Figure 13-21: Relationship Between Primary Grind Size and Mass Pull 163
Figure 13-22: Relationship Between Primary Grind Size and DTR Ni Recovery 164
Figure 13-23: Relationship Between Primary Grind Size and Mass Pull 164
Figure 13-24: Relationship Between Magnetic Field Strength and DTR Ni Recovery 165
Figure 13-25: Relationship Between Magnetic Field Strength and Mass Pull 165
Figure 13-26: Relationship Between Mass Pull and DTR Ni Recovery 165
Figure 13-27: Impact of Grind Size and Magnetic Field Strength on the Relationship Between DTR Ni and DTR Fe Recovery 165
Figure 13-28: Relationship Between Grind Size, Field Strength, and Preferential Grinding on DTR Ni Recovery 168
Figure 13-29: Relationship Between Grind Size, Field Strength, and Preferential Grinding on Tailings Grades 168
Figure 13-30: Relationship Between Grind Size, Field Strength, and Preferential Grinding on Mass Pull 169
Figure 13-31: Relationship Between Mass Pull and DTR Ni Recovery 169
Figure 13-32: Relationship Between Grind Size, Field Strength, and Preferential Grinding On Mass Pull 170

Figure 13-33: Relationship Between Head DTR Fe Grade and Mass Pull 170
Figure 13-34: Cumulative Distribution of DTR Ni Assays from Drill Hole Intervals within the PFS Mine Plan, Overlaid with Variability Samples Grades 171
Figure 13-35: Cumulative Distribution of DTR Fe Assays from Drill Hole Intervals within The PFS Mine Plan, Overlaid with Variability Samples Grades 171
Figure 13-36: Comparison of DTR Results Between Metallurgy Laboratory and Geology Laboratory 172
Figure 13-37: Impact of Magnetic Field Strength and Regrind Size on DTR Ni Recovery 173
Figure 13-38: Impact of Magnetic Field Strength and Regrind Size on DTR Fe Recovery 173
Figure 13-39: Impact of Magnetic Field Strength and Regrind Size on Mass Pull 174
Figure 13-40: Impact of Magnetic Field Strength and Regrind Size on Magnetic Concentrate Iron Grade 174
Figure 13-41: Impact of Magnetic Field Strength and Regrind Size on Mass Pull 174
Figure 13-42: Total Nickel Grades Over the Course of the First Regrind Pilot Plant 176
Figure 13-43: Grade-Recovery Curve from Open-Circuit Testing 183
Figure 13-44: Impact of Acid Addition on Metal Recovery in Flotation Tailings Leach 186
Figure 13-45: Mass Balance Flow Diagram 190
Figure 13-46: Yearly DTR Ni Recovery Projection 190
Figure 13-47: Yearly Total Ni Recovery Projection 190
Figure 14-1: Plan Map Showing Drill Locations and Lithology Model, Baptiste Deposit 193
Figure 14-2: Plan View of Modelled Grade Shells and Implicitly Modelled Dikes at 800 m Elevation 194
Figure 14-3: Cross Section A-A’ of Modelled Grade Shells and Implicitly Modelled Dikes 195
Figure 14-4: Box Plots Showing the % DTR Nickel Grade Distribution Within Mineralized Domains 198
Figure 14-5: Frequency Distribution of % DTR Nickel Grade Within Mineralized Domains 198
Figure 14-6: Variogram Models for Different Plunges and Azimuths of Search Windows 201
Figure 14-7: Variogram Models for Different Plunges and Azimuths of Search Windows 203
Figure 14-8: % DTR Nickel Contact Plot of Low-Grade Domain 204
Figure 14-9: % DTR Nickel Contact Plot of Mid-Grade Domain 204
Figure 14-10: % DTR Nickel Contact Plot of High-Grade Domain 205
Figure 14-11: % DTR Nickel Block Grade Distribution in Plan View at 800 m 207
Figure 14-12: Block Grade Distribution on Cross Section A – A’ from Figure 14-11 207
Figure 14-13: Block Grade Distribution on Longitudinal Section B – B’ from Figure 14-11 208
Figure 14-14: Block Grade Distribution for the Baptiste Deposit Sub-Domains High, Mid, Low 209
Figure 14-15: NNE – SSW Trending Swath Plot (X-Direction) 209
Figure 14-16: ESE – WNW Oriented Swath Plot (Y-Direction) 210
Figure 14-17: Elevation Oriented Swath Plot (Z-Direction) 210
Figure 14-18: Plan View of Resource Classification with 2022 Resource Shell 211
Figure 14-19: Grade – Tonnage Curves* Comparing 2018 and 2022 Resource Estimates 214
Figure 15-1: Pit 68 and 27 (RF 1 and 0.59 Respectively) Whittle Shell Outline 218
Figure 15-2: Whittle Pit-by-Pit Graph with Selected Ultimate Pit 219
Figure 15-3: Plan View of Baptiste Ultimate Pit Design 220
Figure 16-1: Pit-by-Pit Run Results 228
Figure 16-2: Pit Shells in Plan View 229

Figure 16-3: Pit Shells in Section View ... 230
Figure 16-4: Pit Optimization Sensitivity Results ... 231
Figure 16-5: Mine Phase 1 Plan ... 233
Figure 16-6: Mine Phase 2 Plan ... 234
Figure 16-7: Mine Phase 3 Plan ... 235
Figure 16-8: Mine Phase 4 Plan ... 236
Figure 16-9: Mine Phase 5 Plan ... 237
Figure 16-10: Mine Phase 6 Plan ... 238
Figure 16-11: Mine Phase 7 Plan ... 239
Figure 16-12: Mine Phase 8 Plan ... 240
Figure 16-13: Baptiste Grade-Tonnage Summary of Measured and Indicated Reserves ... 242
Figure 16-14: Baptiste Mine Production Schedule ... 245
Figure 16-15: Mine Phase Schedule ... 247
Figure 16-16: Plan at End of Year -1 ... 248
Figure 16-17: Plan at End of Year 1 ... 249
Figure 16-18: Plan at End of Year 2 ... 250
Figure 16-19: Plan at End of Year 5 ... 251
Figure 16-20: Year 10 ... 252
Figure 16-21: Plan at End of Year 15 ... 253
Figure 16-22: Plan at End of Year 21 ... 254
Figure 16-23: Plan at End of Year 25 ... 255
Figure 16-24: Loading Fleet Opportunities ... 258
Figure 16-25: Haulage Fleet Quantities ... 259
Figure 17-1: Overall Process Flow Diagram ... 268
Figure 17-2: Processing Plant Overall Layout ... 269
Figure 17-3: Primary Crushing and Coarse Ore Stockpile Layout ... 270
Figure 17-4: Primary Grinding Area Layout ... 272
Figure 17-5: Magnetic Separation and Regrind Area Layout ... 274
Figure 17-6: Flotation Area Layout ... 277
Figure 17-7: Concentrate Dewatering and Loadout Area Layout ... 279
Figure 17-8: Flotation Tailings Leaching Area Layout ... 281
Figure 17-9: Solution Purification and MHP Precipitation Area Layout ... 282
Figure 17-10: Reagent Area Layout ... 284
Figure 17-11: Services and Utilities Area Layout ... 287
Figure 18-1: Overall Site Layout ... 291
Figure 18-2: Off-site Access Road System Overview ... 293
Figure 18-3: Mine Maintenance Facilities ... 295
Figure 22-1: Projected Post-Tax Undiscounted Cash Flow ... 353
Figure 22-2: Pre-Tax NPV 8% Sensitivity Chart (US$M) ... 358
Figure 22-3: Pre-Tax IRR Sensitivity Chart ... 358
Figure 22-4: Post-Tax NPV 8% Sensitivity Chart (US$M) ... 360

Figure 22-5: Post-Tax IRR Sensitivity Chart 360
Figure 23-1: Location of Adjacent Properties 362
Figure 24-1: Summary Execution Schedule 364
Figure 24-2: Lead Consultant EPCM Milestone Dates 367
Figure 24-3: Block Flow Diagram of the Refinery Option Processing Route 372
Figure 24-4: Overall Process Flow Diagram – Refinery Option 384
Figure 24-5: Overall Site Layout – Refinery Option 385
Figure 24-6: PFS Base Case + Refinery Option Scenario: Post-Tax NPV 8% Sensitivity Chart (US$M) 410
Figure 24-7: PFS Base Case + Refinery Option Scenario: Post-Tax IRR Sensitivity Chart 410

Appendices

Appendix A – MINERAL CLAIMS 434

1 SUMMARY

1.1 Introduction

The Baptiste Nickel Project (Baptiste, or the Project) is a greenfield project in the Nechako region of central British Columbia, located approximately 90 km northwest of Fort St. James. The Project is entirely located within the Decar Property (Decar) claims package, which is 100% owned by FPX Nickel Corp. (FPX).

The ultramafic resource is both nickel-rich and sulphur-deficient, which has resulted in an advantageous form of nickel mineralization, awaruite (Ni_3Fe). The resource is both large and homogeneous, lending itself to large-scale open pit mining methods. The on-site mineral processing plant will utilize well proven unit operations including semi-autogenous grinding (SAG) milling, ball milling, magnetic separation, regrinding, froth flotation, and mild tails leaching. Off-site linear infrastructure includes upgrades to an existing access road network, connection to the BC Hydro electrical grid, and a fresh water pipeline system.

This prefeasibility study (PFS) incorporates the results of FPX's extensive de-risking program, which has advanced project maturity and understanding following the 2020 preliminary economic assessment (PEA). Key elements included in this PFS include:

- Geology: Inclusion of results from the 2021 infill drilling program and a more representative modelling approach.
- Metallurgy: Inclusion of results from multiple bench- and pilot-scale programs.
- Engineering: Inclusion of results from a series of trade-off studies evaluating project scale, project phasing, mine planning, comminution technology, primary grind size, regrind configuration, tailings leach circuit, water supply source, off-site power connection location, access road alignment, and tailings facility.
- Field Investigations: Inclusion of results from two seasons of geotechnical site investigations and 18 months of cultural and environmental baseline studies.

The PFS Base Case (Base Case) considers a phased project approach with a 29-year mine life. Phase 1 includes the first nine years of operation and has a mining and processing rate of 108 kt/d of ore. Phase 2 commences at the start of Year 10 with the commissioning of a plant expansion to bring the total processing rates to 162 kt/d of ore. The operation will produce two nickel coproducts, with approximately 93% of contained nickel in a high-grade (60% nickel) awaruite concentrate, and the balance in a mixed hydroxide precipitate (MHP) product. For the Base Case, the concentrate will be marketed directly to stainless steel producers achieving ferronickel (FeNi) type payabilities, which is possible without any need for intermediate smelting. For the base case, the MHP coproduct will be marketed to the EV battery material supply chain.

This PFS also presents an option for an off-site refinery to produce 40 kt/a of nickel in battery-grade nickel sulphate, along with cobalt and copper byproducts (Refinery Option). The nickel sulphate and cobalt product will be marketed to the EV battery material supply chain, while the copper product will be marketed to the traditional copper supply chain. Any nickel in excess of the 40 kt/a in nickel sulphate will continue to be marketed directly to stainless steel producers.

This refinery concept is presented as a discrete option in Section 24 of this report. All other sections of this report pertain to the PFS Base Case exclusively.

FPX commissioned Ausenco Engineering Canada Inc. (Ausenco) to compile the PFS for the Baptiste Nickel Project. The PFS was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1.

The responsibilities of the engineering companies who were contracted by FPX to prepare this report are as follows:

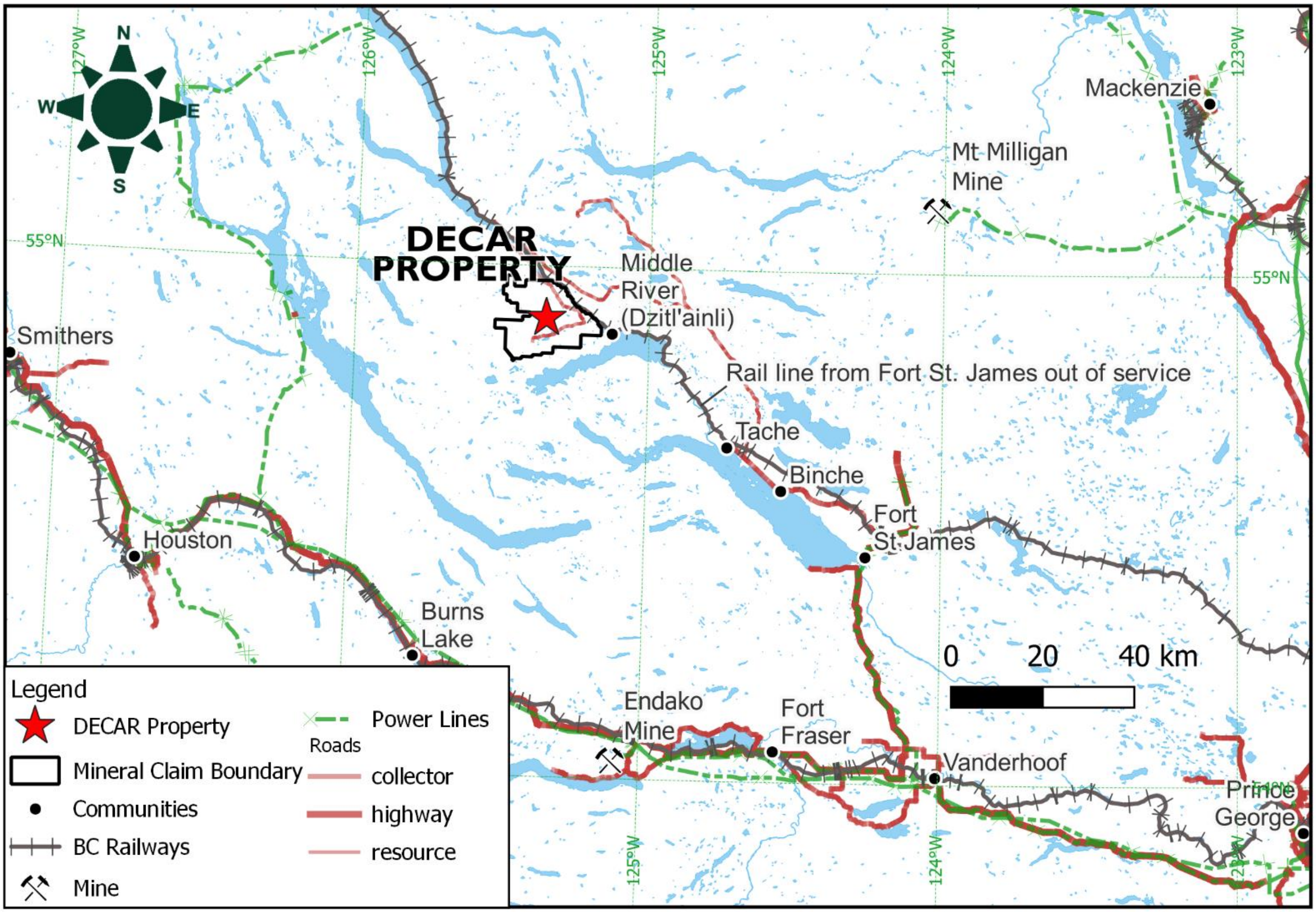
- Ausenco managed and coordinated the development of this report, developed PFS-level designs for the process plant and general on-site infrastructure, developed the consolidated cost estimates, and prepared the economic analysis.
- Carisbrooke Consulting Inc. (Carisbrooke) provided the off-site power infrastructure design and cost estimate.
- Equity Exploration Consultants Ltd. (Equity Exploration) completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, exploration works, sample preparation and analysis, and data verification.
- ERM Consultants Canada Ltd. (ERM) conducted a review of the environmental studies and permitting information.
- International Metallurgical and Environmental Inc. (IMI) coordinated and reviewed the metallurgical test results and defined process design criteria.
- Knight Piésold Ltd. (KP) supervised geotechnical investigations, provided geotechnical recommendations, and developed the PFS-level design and cost estimate of the tailings and water management facilities.
- Next Mine Consulting Ltd. (Next Mine) developed the mineral resource estimate.
- Onsite Engineering Ltd. (Onsite Engineering) prepared the design and cost estimate for the off-site road system.
- TechSer Mining Consultants Ltd. (TechSer) designed the open pit mine, mine production schedule, and mine capital and operating costs.

1.2 Property Description, Location and Ownership

The Decar Property comprises 62 contiguous mineral claims that cover 24,740 hectares (247 km^2) in the Omineca Mining Division of central British Columbia, Canada (Figure 1-1). The claims confer title to subsurface mineral tenure only, as defined by the Mineral Tenure Act of BC. Surface rights over non-overlapping MTO claims are held by the Crown, as administered by the Government of BC. The ownership of other rights (placer, timber, water, grazing, trapping, outfitting, etc.) affecting the property was not investigated by the authors.

MTO claim data for the Decar property is summarized in Figure 1-2. There are some overlaps with a Provincial Park, Mineral Reserve, and legacy claims that slightly reduce the effective size of the property, but these are all at least 2.5 km away from known awaruite deposits and targets. As of August 2023, all 62 of the Decar Property claims are in good standing until at least July 2032.

Figure 1-1: Location Map for the Decar Property



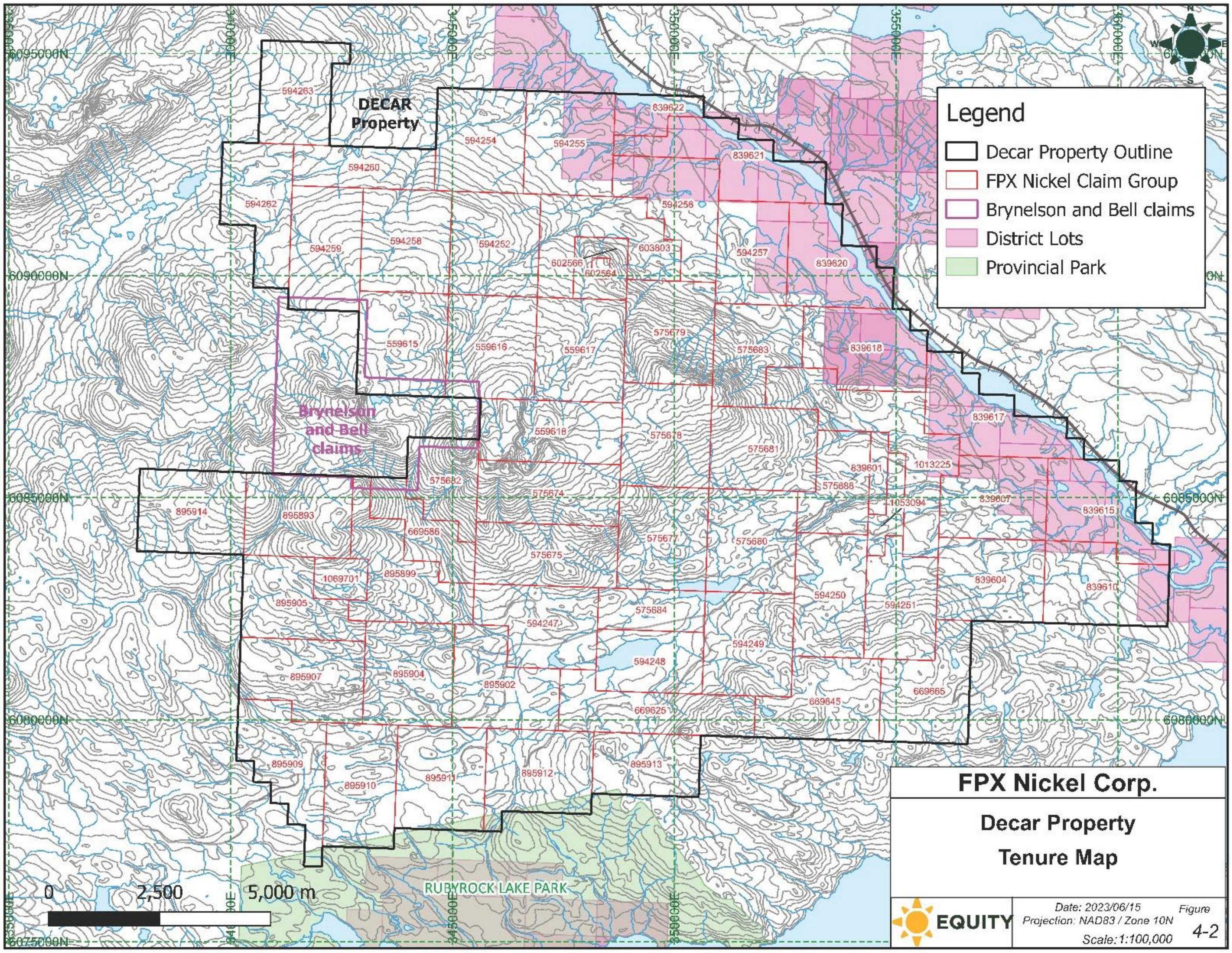
Source: Equity Exploration, 2023

All claims are registered in the name of and are 100% owned by FPX and are subject to a 1% net smelter return (NSR) royalty. The authors are otherwise unaware of any other royalties, back-in rights, or other agreements and encumbrances to which the property is subject.

No material environmental liabilities were noted by authors Voordouw and Flynn during a visit to the Decar Property in July 2022. The authors are otherwise unaware of any environmental liabilities or any other risks that may prevent FPX from carrying out future work.

To the authors' knowledge, there are no other significant factors and risks that may affect access, title, or the right, or ability to perform work on the property.

Figure 1-2: Decar Property Tenure Map



Note: A list of claims is provided in Appendix A.
Source: Equity Exploration, 2023

1.3 History

Exploration activity on or immediately adjacent to the Decar Property focused mostly on chromite and platinum group elements (PGE) from the mid-1970s to mid-1980s and listwanite-hosted gold from 1987 to 1994. The strongly magnetic and high-density awaruite was first identified in 1983 but did not become a target for mineral exploration until the mid-1990s.

In 1996, geochemical and petrographic work on ultramafic samples from what is now the centre of the Decar Property showed nickel hosted in awaruite and other low-sulphur nickel minerals, including heazlewoodite, bravoite and pentlandite. It was recognized that since awaruite, magnetite and chromite were all magnetic, a concentrate could be produced by magnetic separation. These results were followed up with sporadic rock sampling and metallurgical work till 2007 when First Point Minerals Corp. (First Point), the precursor to FPX, staked 15 claims that now form the core of the Decar Property. FPX then steadily grew the Decar Property to its current 62-claim size by 2018.

In 2009, FPX optioned the Property to Cliffs Natural Resources Inc. (Cliffs), who earned a 60% interest in the property by incurring approximately US$22 million in exploration and related expenditures that included completion of the first preliminary economic assessment (PEA) in 2013. In 2015, FPX re-acquired Cliffs' 60% ownership interest with a cash payment of US$4.75 M, provided by a Lender who earned a 1% NSR royalty over the Decar Project, which has since been fully repaid.

Historical mineral resource estimates were completed by FPX in 2012, 2013, 2017, and 2020. All four of these estimates were completed in accordance with NI 43-101 reporting standards and were calculated for a cut-off grade of 0.06% DTR Ni. These historical resource estimates have not been verified by the authors, are not considered relevant, and should not be relied upon for any use. A qualified person has not done sufficient work to classify these historical estimates as current mineral resources and FPX is not treating any of the historical estimates as current mineral resources. An updated resource for the project is provided in Section 14 of this technical report.

No other significant historical mineral resource estimates have been reported for the property.

1.4 Geology and Mineralization

The Decar Property lies entirely within the Cache Creek terrane of British Columbia, with awaruite mineralization hosted by the Trembleur ultramafite unit and correlated to increased serpentinization. This alteration is best developed in association with NW-striking shear zones defined by a core of cataclasite, mylonite, and/or fault breccia that passes symmetrically outwards into decreasing intensities of penetrative deformation fabric. The Baptiste Deposit, Sid target, and Van target, for example, are all elongated along a NW-SE direction and bound by subvertical, NW-SE striking, 50-100 m wide, belts of strong penetrative deformation fabric. This relationship suggests that NW-striking structures channeled the fluids that caused serpentinization of the Trembleur ultramafite and resultant awaruite mineralization. The surface expression of the Baptiste Deposit also suggests a possible secondary NE-directed control.

Serpentinization and Mg-Fe carbonate alteration are the predominant alteration types within the Trembleur ultramafite. Serpentinization is the most widespread, with all ultramafic rocks within the Trembleur unit estimated to be 60-100% serpentinized. On the Decar Property, serpentinization is defined by the replacement of olivine and orthopyroxene with antigorite and lizardite, with magnetite and awaruite forming as part of the serpentinization process.

Awaruite deposits are formed by the serpentinization of magmatic olivine that leads to the liberation of nickel and iron and subsequent formation of awaruite. Awaruite occurs as disseminated grains throughout the entire extent of the peridotite on the Decar Property, but four zones of more abundant mineralization and larger grain size have been identified and named: Baptiste, Sid, B, and Van. The Baptiste Deposit and Van target are the most advanced of these zones, with both tested by numerous drill holes (see Section 10). The Sid and B targets are defined through surface mapping, sampling and/or diamond drilling, with three holes drilled at Sid target and one drilled at B target. High-grade awaruite zones trend NW-SE, parallel to lithological contacts and older fault structures.

The Baptiste Deposit is currently defined for approximately 3,000 m in a west of northwest to east of southeast direction, and for 400-1,500 m from north-northeast to south-southwest. Diamond drilling shows that mineralization typically extends to vertical depths of at least 200-300 m, with 13 of 28 holes drilled deeper than 300 m ending in mineralization. The Deposit is in fault contact with Sitlika metavolcanic rocks to the southwest and grades into massive peridotite with lower-grade mineralization to the north and northwest.

1.5 Exploration

FPX staked approximately half of what is currently the Decar Property in 2006 and then conducted four years of additional staking, geological mapping and sampling, geophysical surveys, and metallurgical testwork before optioning the project to Cliffs in 2009. Cliffs then funded exploration on the Property from 2010 to 2013, publishing a maiden resource for the Baptiste Deposit in 2012 followed by an updated resource and PEA in 2013. Since 2013, FPX has completed additional mapping and surface sampling work in addition to diamond drilling in 2017 and 2021.

Total exploration work completed by FPX and its predecessor, First Point, include 1,668 line-km of airborne magnetic survey, 29.1 line-km of ground-based induced polarization and resistivity (IP) surveys, 45 line-km of ground-based magnetic surveys, downhole geophysical surveys and physical property work, and collection of 1,206 rock samples as well as 111 stream sediments. This work has shown that magnetic surveys and rock sampling are effective methods for delineating awaruite mineralization.

1.6 Drilling

Diamond drilling for awaruite deposits on the Decar Property was initiated in 2010 with subsequent campaigns in 2011, 2012, 2017, 2021, and 2022, for a total of 40,652 m drilled in 131 holes (Table 1-1). The bulk of these meters were allocated to the Baptiste Deposit and Van target, whereas other drill-tested targets include B and Sid. For Baptiste, most holes were drilled at dips of -50° to -60° through vertically oriented mineralization, so that the horizontal and vertical extents of the mineralization is equal to 50-65% and 75-85%, respectively, of their downhole length.

Approximately 60% of holes drilled in the Baptiste area returned composites >20 DTR Ni %*m (i.e., equivalent to 200 m at 0.1% DTR Ni) with the remaining 40% of holes mostly defining the periphery of the deposit. Likewise, drilling at the Van target returned approximately 50% of holes with >20 DTR Ni %*m, whereas the one hole drilled at B target and two of three drilled at Sid also returned >20 DTR Ni %*m.

Table 1-1: Summary of Drilling Completed by FPX on the Decar Property

Year	Target	DDH (quantity)	DDH (Total m)	Average m/DDH
2010	Baptiste	7	1,711	244
	Sid	3	847	282
2011	Baptiste	35	10,864	310
	B target	1	305	305
2012	Baptiste	27	15,096	559
	Baptiste Engineering	7	1,401	200
2017	Baptiste	8	1,918	240
2021	Baptiste	17	2,856	168
	Van	9	2,688	299
2022	Van	10	2504	250
	Baptiste Engineering	7	463	66
Total		**131**	**40,652**	**310**

1.7 Sample Preparation, Analysis and Security

Drill core sampling for the Decar Property was completed in seven campaigns, with six of these done concurrently with drilling, and one consisting of infill sampling programs of 2010 and 2011 drill core in 2012 (Table 1-2).

Table 1-2: Overview of Decar Drill Core Sampling Campaigns

Drill Campaign	Sampling Campaign	Core Samples Assayed			
		Quantity	Total Length (m)	Average Length (m)	% of Length
2010	2010 drilling	454	466.9	1.0	18%
	2012 resampling	345	1,272.4	3.7	50%
2011	2011 drilling	2,620	2,618.8	1.0	23%
	2012 resampling	2,600	7,578.7	2.9	68%
2012	2012 drilling	4,192	15,552.3	3.7	94%
2017	2017 drilling	460	1,565.1	3.4	82%
2021	2021 drilling	1,384	4,842.1	3.5	87%
2022	2022 drilling	657	2,346.7	3.6	79%
Total		**12,712**	**36,243.0**	**2.9**	**89%**

On-site sample preparation procedures were similar for all sampling campaigns. In each case, the selected drill core intervals were cut with a core saw, with one half placed into a plastic bag with a pre-numbered sample tag (core sample) and the other half returned to the core box for reference (as reference core). Standards (CRM, non-certified standards, replicates) and blank samples were alternately inserted as every 10th sample for a total insertion rate of approximately 10%. One core (quartered core) duplicate was also inserted in every 20 samples for the 2010, 2011, 2012, and 2017 programs; this was reduced to one in every 40 samples for the 2021 and 2022 work. Coarse core or preparation duplicates were also inserted at a rate of 1 in every 20 samples for the 2010, 2011, 2012, and 2017 programs but were dropped from the 2021 and 2022 work.

All drill core from the Decar property has been transported to Fort St. James, BC, where it is stored in a fenced compound owned by Russell Transfer Ltd. Core boxes are stored in racks and are generally well preserved as of the June 2022 site visit (see Section 12).

All core samples have been analyzed twice, with one analysis on a whole rock pulp and another on a magnetic separate generated with a Davis Tube magnetic separator. The initial preparation stage is the same, however, with the entire core sample crushed and then split into a 250-gram sub-sample that is pulverized to a pulp with 95% of particles passing 75 µm (200-mesh). For whole rock analyses, a portion of the sub-sample is fused with lithium metaborate/tetraborate flux and analysed by inductively coupled plasma optical emission spectrometry (ICP-OES).

The magnetic separate sample is generated by running the pulverized sub-sample through a Davis Tube magnetic separator, which splits the sub-sample into magnetic and non-magnetic fractions. The magnetic fraction is then fused with lithium metaborate/tetraborate flux and analysed by X-ray fluorescence (XRF). These analyses are more representative of the minable grade of the deposit since most recoverable nickel will be in the magnetic separate.

Quality assurance / quality control (QA/QC) of geochemical analyses was monitored with certified reference materials (CRMs), non-certified standards (NCS), replicates, blanks, and duplicates. The core cutting, bagging, and transport procedures for all programs are industry standard. The authors are unaware of any security concerns related to drill core from the 2010, 2011, 2012, 2017, 2021 or 2022 programs. It is the QP's opinion that the methods used to split, sample, secure, and transport drill core are industry standard, and that the QA/QC procedures are in line with industry best practise. QA/QC sample insertion rates exceed industry best practice. The analytical methods and QA/QC results are also considered adequate.

1.8 Data Verification

Data verification done by QP Flynn included:

- a visit to the core yard at Fort St James and a visit to the Decar Property, including the Baptiste Deposit, from July 4th to 7th, 2022;
- 50 spot checks of assays in the database against lab certificates;
- field duplicate assays of 10 assay intervals; and
- a review of QA/QC data compiled by QP Voordouw in Section 11.

The core yard work included a review of zones with between 0.04% to 0.23% DTR Ni from drill holes 21BAP084, 073, 085, and 087. The majority of the higher grade mineralized intervals included visible awaruite.

The 50 spot checks returned no inconsistencies between the original certificate of analysis and the values entered into FPX's database. Field duplicates showed adequate reproducibility of original assay results.

In the September 2018 site visit by QP Voordouw, several outcrops examined at the northwestern end of the Baptiste Deposit were found to consist of peridotite and dunite. Other outcrops found along the road include argillite and mafic volcanic of the Sitlika assemblage, the distribution of which is consistent with property-scale mapping.

The 2018 and 2022 site visits and review of FPX data and management of the 2017 and 2021 drill programs by Equity Exploration have provided the necessary level of confidence that data from the Decar Project is adequate for the purposes of this report.

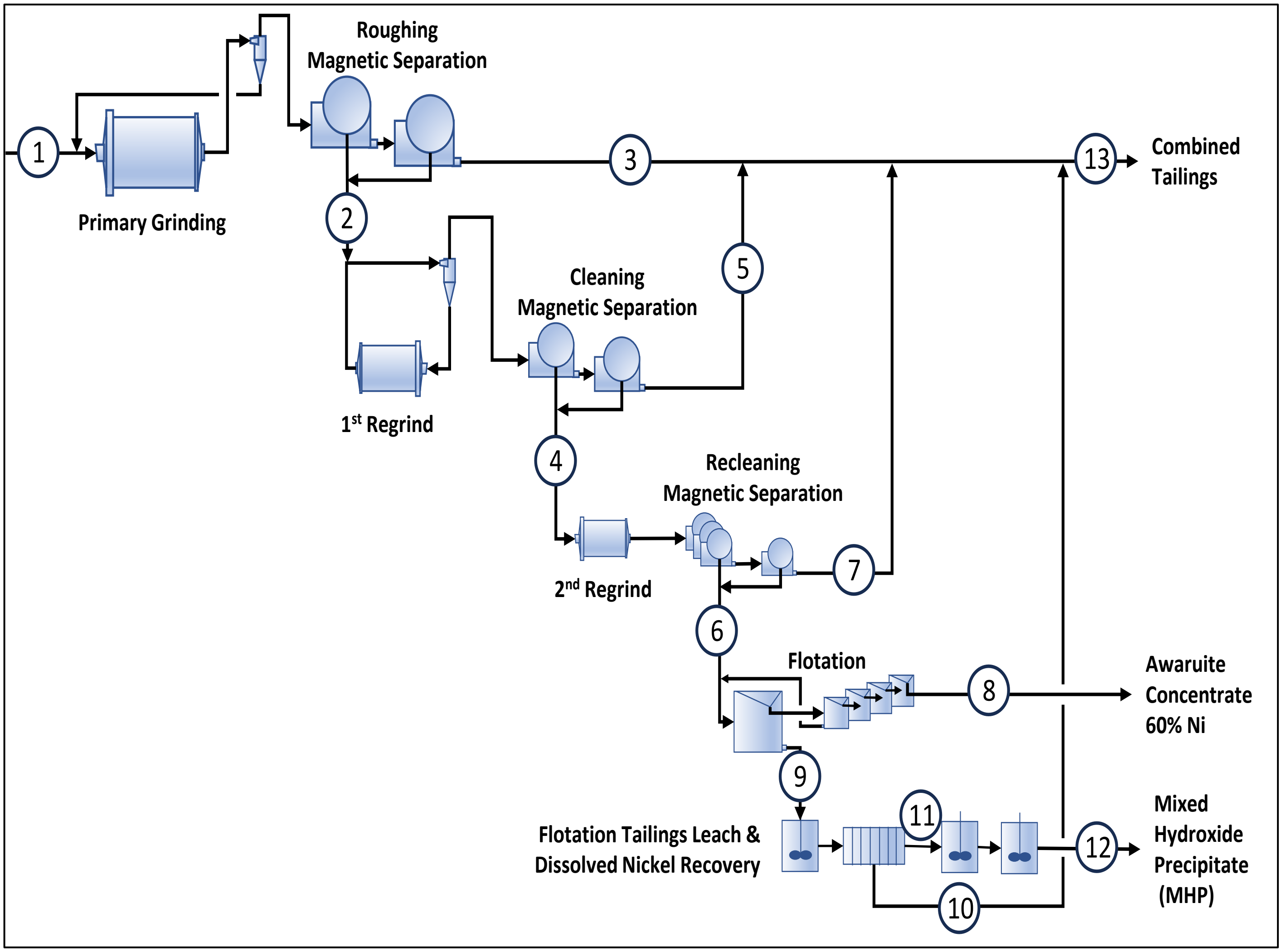
1.9 Mineral Processing and Metallurgical Testwork

The understanding of the metallurgical process requirements for the recovery of nickel from the Baptiste Deposit has evolved during approximately 14 years of metallurgical testing at various facilities using several representative samples. Nickel occurs primarily as fine and very fine grained awaruite particles in a serpentine host rock.

From the outset of testing, there has been a consistent reliance on magnetic processes for the recovery of awaruite and magnetite in a cost-effective rougher stage. The upgrading of nickel from the rougher stage requires regrinding of rougher concentrates to fine size distributions and includes several stages of magnetic upgrading to produce a magnetic concentrate that is predominantly magnetite and awaruite. This magnetic concentrate can be used as feed to a flotation process to separate and recover nickel to a high-grade nickel concentrate. Minor nickel losses in the flotation tailings can be partially recovered using a simple atmospheric leach process and precipitation process. The overall process flowsheet is shown in Figure 1-3 below.

The process relies on several key mineralogical characteristics of the Baptiste Deposit. Firstly, the nickel mineral awaruite is easily recovered using magnetic separation and this allows for the recovery of awaruite, even in poorly liberated particles. This characteristic allows for the use of a relatively coarse primary grind (250 µm) in the preliminary stages of nickel recovery. Secondly, the grain sizes observed for awaruite are very fine, and consistently in the range of 5-10 µm and this dictates that fine-grinding (15-25 µm) is required to properly liberate awaruite and allow for the production of high-grade nickel concentrates using flotation.

Figure 1-3: Baptiste Concentrator Block Flow Diagram



Source: FPX, 2023.

A significant variability program was completed in 2023 which demonstrated the consistent metallurgical response of the Baptiste materials to the proposed process flowsheet.

As a low-grade, high-tonnage project, the Baptiste testwork programs relied on low-cost magnetic separation technology. The development of a successful flotation process to generate high-value magnetic nickel concentrates has added significant value to the project and places the Baptiste Deposit in a unique marketing position. Saleable concentrates of approximately 60% nickel are expected once the project is operating.

Testwork using a life-of-mine (LOM) composite sample from the Baptiste Deposit has been completed and is summarized in Table 1-3. This metallurgical result is consistent with other samples tested within the variability test work program. Nickel values are accounted for using both a DTR methodology and a traditional total nickel methodology and the differences reflects the magnetic responses of different nickel minerals within the Baptiste Deposit.

Table 1-3: Metallurgical Results for Life-of-Mine Composite Sample

	Overall Distribution (%)				Grades (%)		
	Mass	DTR Ni	Total Ni	DTR Fe	DTR Ni	Total Ni	DTR Fe
Mill Feed	100	100	100	100	0.13	0.21	2.39
Magnetic Concentrate	4.3	94.1	58.2	94.1	2.83	2.83	52.0
Non-magnetic Tailings	95.7	5.9	41.8	5.9	0.008	0.09	0.15
Flotation Concentrate	0.18	82.2	50.9	1.8	60.0	60.0	24.6
Flotation Tailings	4.1	8.92	5.5	92.2	0.28	0.28	53.2
Mixed Hydroxide Precipitate	0.02	6.5	4.0	0	45	45	0
Leach Tailings	4.1	4.5	2.8	92.2	0.14	0.14	53.8
Combined Tailings	99.8	11.3	45.1	98.2	0.015	0.09	2.35

Note: Abbreviated mass balance shown. Refer to Table 13-16 for complete balance.

1.10 Mineral Resource Estimate

The sample database supplied for the Baptiste Deposit contains results from 99 surface drillholes completed since 2010 (Table 14-1), or 96% of all metres drilled as shown in Table 10-1. In comparison to the 2020 Resource Estimate, the 2022 Resource Estimate incorporated an additional 17 diamond drillholes totalling 2,856 m from the 2021 drilling campaign. The average drillhole spacing in the Baptiste Deposit is 150 m.

The 2022 updated geological interpretation was provided by Equity Exploration as a model of lithology (Figure 14-1). The model was created using Leapfrog Geo using implicit modelling methods. The updated geological model of the Baptiste Deposit was drafted by Equity Exploration and consists of four mineralized and six unmineralized (or barren) domains (Figure 14-2). The mineralized domains are:

- Peridotite - Mineralized: main host of nickel mineralization, logged as “peridotite”, includes variably brecciated cataclastic and/or mylonitized peridotite.
- Dunite: grade is variable but typically lower than peridotite – mineralized, logged as “dunite”, forms layers in Peridotite - mineralized domain, marked by relatively fine grain size, locally brecciated.
- Peridotite - Massive: weakly mineralized, logged as “massive peridotite” or possibly “peridotite”, forms transition from peridotite - mineralized to - low-grade domains, more vein-controlled serpentinization as opposed to pervasive serpentinite alteration in peridotite - mineralized domain.
- Fe-Carbonate Altered Ultramafic: weakly mineralized, logged as “FeCb_AltUM”, weak to moderate (incipient) Fe-carbonate altered peridotite, forms gradation from Peridotite - Mineralized to barren Listwanite and Fe-Cb altered domains, occurs mostly along southern margin of Baptiste Deposit and around the Listwanite ellipse bounding most of the eastern margin.

Sub-domains (DTR Ni grade shells) of the provided lithological model were modeled in Leapfrog. DTR Nickel grades within the corresponding domains were estimated in two passes using ordinary kriging (OK) for grade shells and inverse distance squared (ID2) for the dikes. Model validation was done visually and using swath plots. The anisotropy conforms to the search ellipsoids derived from the variogram models. Locally varying anisotropy was used for all grade shell and dike domains. The dikes employed the central trend line derived from the Leapfrog 3D vein models. The grade shells utilized concentric patterns that honoured the shape of their 'nested' contacts and that samples should be paired as such for estimation. The high-grade shell also used the major axis of the variogram model.

Mineral resources were constrained by an optimized pit shell based on an exchange rate of C$1 = US$0.77 and a nickel price of US$8.50/lb with 96% payability. Total operating costs, including mining, processing, G&A, and minimum profit were US$9.37/t milled. A mining recovery of 95% and process recovery of 85% were used in the optimization. A base case cut-off grade of 0.06% DTR Nickel represents an in-situ metal value of approximately US$9/t. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-4 provides the summary of the 2022 Baptiste Deposit pit-constrained mineral resource estimate.

Table 1-4: 2022 Baptiste Deposit Pit-Constrained Mineral Resource Estimate

Category	Tonnes (Mt)	DTR Ni content		Total Ni content		DTR Fe content		DTR Co content	
		(% Ni)	(kt Ni)	(% Ni)	(kt Ni)	(% Fe)	(kt Fe)	(% Co)	(kt Co)
Indicated	1,815	0.129	2,435	0.211	3,828	2.40	43,535	0.0035	64.4
Inferred	339	0.131	2,436	0.212	720	2.55	8,634	0.0037	12.5

Notes:
1. Mineral resource estimate prepared by Richard Flynn, P.Geo of Next Mine Consulting using ordinary kriging within grade shell domains and inverse distance squared in dike domains; effective date of November 14, 2022.
2. Resources are reported using the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines.
3. Davis Tube magnetically-recovered (DTR) nickel is the nickel content recovered by magnetic separation using a Davis Tube, followed by Fusion-XRF to determine the nickel content of the magnetic fraction; in effect a mini-scale metallurgical test. The Davis Tube method is the global, industry standard metallurgical testing apparatus for recovery of magnetic minerals.
4. Indicated mineral resources are drilled on approximately 200 x 200 m drill spacing and confined to mineralized lithologic domains. Inferred mineral resources are drilled on 300 x 300-metre drill spacing.
5. A cut-off grade of 0.06% DTR Ni was applied.
6. An optimized pit shell was generated using the following assumptions: US$8.50/lb nickel price; pit slopes between 42-44°; nickel payability of 96%, mining recovery of 97% DTR Ni, process recovery of 85% DTR Ni, exchange rate of US$1.00=C $0.77; and total operating cost and minimum profit of US$9.37/t milled.
7. Totals may not sum due to rounding.
8. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. Table 1-5 presents the mineral resource estimate for the Baptiste Deposit at a range of cut-off grades with the applied cut-off grade of 0.06% DTR Nickel in bold face.

Table 1-5: Indicated and Inferred Resources for the Baptiste Deposit

Indicated			
% DTR Ni Cut-off	**Tonnes (Mt)**	**DTR Ni (%)**	**Contained Ni (kt)**
0.04	1,850	0.128	2,364
0.06	**1,815**	**0.129**	**2,346**
0.08	1,730	0.132	2,285
0.10	1,576	0.136	2,144
Inferred			
% DTR Ni Cut-off	**Tonnes (Mt)**	**DTR Ni (%)**	**Contained Ni (kt)**
0.04	351	0.128	449
0.06	**339**	**0.131**	**443**
0.08	319	0.134	428
0.10	288	0.139	400

Notes:
1. Mineral resource estimate prepared by Richard Flynn, P.Geo of Next Mine Consulting using ordinary kriging within grade shell domains and inverse distance squared in dike domains; effective date of November 14, 2022.
2. Resources are reported using the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines.
3. Davis Tube magnetically-recovered (DTR) nickel is the nickel content recovered by magnetic separation using a Davis Tube, followed by Fusion XRF to determine the nickel content of the magnetic fraction; in effect a mini-scale metallurgical test. The Davis Tube method is the global, industry standard metallurgical testing apparatus for recovery of magnetic minerals.
4. Indicated mineral resources are drilled on approximately 200 x 200 m drill spacing and confined to mineralized lithologic domains. Inferred mineral resources are drilled on approximately 300 x 300 m drill spacing.
5. A cut-off grade of 0.06% DTR Ni was applied.
6. An optimized pit shell was generated using the following assumptions: US$8.50 per pound nickel Price; pit slopes between 42-44°; nickel payability of 96%, mining recovery of 97% DTR Ni, process recovery of 85% DTR Ni, exchange rate of US$1.00=C $0.77; and total operating cost and minimum profit of US$9.37/t milled.
7. Totals may not sum due to rounding.
8. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

1.11 Mineral Reserve Statement

The Baptiste Deposit is a large, near surface, bulk mineable deposit that is well suited for conventional large-scale truck and shovel operation. The Mineral Reserves for Baptiste were based on the Project's Mineral Resource estimate described in Section 14 of this report. Only DTR nickel grades were used for net smelter return (NSR) calculations. Only Measured and Indicated Mineral Resources were considered for processing, with Inferred Mineral Resources treated as waste.

The estimated Mineral Reserves are reported using a metal price of $8.75/lb Ni and cut-off grade of 0.06% DTR nickel, which is equivalent to approximately US$ 9/t contained metal value. This results in an average NSR of $19.80/t over the planned life of mine (LOM). The Baptiste Reserve Estimate is summarized in Table 1-6.

Table 1-6: Baptiste Nickel Project Reserve Estimate

Category	Tonnes (kt)	DTR Ni (%)	Total Ni (%)	Contained Metal, DTR Ni (kt)	Contained Metal, Total Ni (kt)
Proven	-	-	-	-	-
Probable	1,488	0.130	0.210	1,933	3,125
Total Proven and Probable	**1,488**	**0.130**	**0.210**	**1,933**	**3,125**

Notes:

1. Mineral Reserves are reported effective September 6, 2023. The Qualified Person for the estimate is Mr. Cristian Hernan Garcia Jimenez, P.Eng, an independent consultant.
2. Mineral Reserves are reported using a fixed 0.06% Ni DTR cut-off grade, which represent approximately US$9.00/t NSR value, which is above the economic cut-off grade of US$5.5/t.
3. The Mineral Reserves are supported by a mine plan, based on a pit design, guided by a Lerchs Grossmann (LG) pit shell. Inputs to that process are:
   - Metal prices of Ni US$8.75/lb.
   - Mining cost US$1.98 /t moved.
   - An average processing cost of US$3.72 /t milled, which includes processing and tailing storage costs.
   - General and administrative cost of US$1.10 /t milled.
   - Pit overall slope angles varying from 42 to 44 degrees.
   - A fixed process recovery of 85% for all the measured and indicated blocks above 0.06% DTR nickel grade.
4. The life-of-mine strip ratio is 0.56 (W:O), excluding capitalized pre-stripping.
5. Tonnage and contained Nickel tonnes are reported in metric units and grades are reported as percentages.
6. All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.

Mineral Reserves are reported effective September 6, 2023, using CIM Definition Standards. The Qualified Person for the estimate is Mr. Cristian Hernan Garcia Jimenez, P.Eng., independent consultant.

1.12 Mining Methods

The Baptiste Deposit will be mined via bulk open pit mining operations that will be developed in two phases. Phase 1 includes the first 9 years of operation (plus the preceding 2 years of mine pre-stripping) with a nominal ore processing rate of 108 kt/d. Phase 2 begins in Operating Year 10, is 19 years in duration, and coincides with a plant expansion to a nominal ore processing rate of 162 kt/d.

The Baptiste Deposit will be mined as a conventional large-scale truck and shovel operation with up to 60 Mt of material mined per annum during Phase 1, and up to 120 Mt of material mined per annum during Phase 2. The mining operation will feature 250 mm blast hole electric drills, 42 m^3 electric excavators, and 300 t class haul trucks working on nominal 10-m high benches. A flexible combination of dozers, graders, wheel loaders, and excavators will form the core of the support equipment fleet.

Two years of pre-production mining are required to carry out the following tasks:

- Pioneering: This includes the development of 13 km of mining roads which are mostly constructed of fill from overburden or waste rock. The roads will connect the upper elevation of the Mine Phase 1 pit to the tailings facility starter dams, the overburden stockpile, the operational ore stockpile, and the primary crusher dump area. This work will also create a platform for Mine Phase 1 and Mine Phase 2 which will be sufficient to allow the operation of the mining equipment.

- Pre-Stripping: This includes the stripping of 38.6 Mt of waste rock from Mine Phase 1, which will expose sufficient ore to allow continuous ore delivery to the concentrator upon start-up. During this period, 14.8 Mt of ore will be transported to the operational ore stockpile for temporary storage.

Over the project's 29-year life, the concentrator will be fed with ore directly from the mine and the operational ore stockpile. The total production for the first year is 29.6 Mt (82 kt/d), which represents 75% of the concentrator's nameplate capacity. Throughput is restricted this year as the plant commissioning is completed and throughput is ramped-up. In Years 2 through 9, the full 39.4 Mt (108 kt/d) of ore is delivered to the primary crusher. During this period, the mine ramps-up from approximately 15 Mt in Year -2 up to 50 Mt in Year 3. Then, from Year 4 through 9 the mine moves an average of 60 Mt per annum. In Year 10, a third grinding line will be commissioned, and the average annual concentrator throughput will increase to 59.1 Mt per annum (162 kt/d). During Years 10 through 29, the mine reaches a peak mining capacity of 120 Mt per annum, with an average mine capacity of 96 Mt per annum.

Table 1-7 presents tonnages and grades for all material within the eight mining phases designed for the Baptiste PFS. The data is based on the selected cut-off for the project (0.06% DTR Ni). Note that in the optimization and design, all Inferred material has been considered as waste. With further drilling there is the potential for upgrading this material to Indicated or Measured status, which could contribute an additional 44 Mt of ore to the Project.

Table 1-7: Summary of Mine Phase Tonnages and Grades

Phase	Ore (kt)	DTR Ni (%)	DTR Co (%)	DTR Fe (%)	NSR ($/t)[1]	Waste (kt)	Inferred (kt)	Total (kt)
Phase 1	131,485	0.142	0.004	2.8	21.6	42,888	134	174,507
Phase 2	132,461	0.137	0.004	2.7	20.8	70,827	0	203,287
Phase 3	170,050	0.133	0.004	2.5	20.2	43,876	3,156	217,081
Phase 4	221,215	0.135	0.004	2.6	20.5	169,365	1,847	392,428
Phase 5	225,899	0.131	0.003	2.5	19.9	113,860	12,478	352,237
Phase 6	256,899	0.120	0.003	1.9	18.3	205,239	6,228	468,365
Phase 7	233,329	0.129	0.003	2.0	19.6	127,972	12,125	373,426
Phase 8	116,384	0.117	0.003	2.5	17.8	57,943	8,254	182,581
Total	**1,487,721**	**0.130**	**0.004**	**2.39**	**19.765**	**831,970**	**44,221**	**2,363,912**

[1] Considers only DTR Ni metal value.

Based on the homogeneity of the Baptiste Deposit and the utilization of an elevated cut-off grade (0.06% DTR Ni) during pit optimization and mine planning, it is reasonable to state the occurrences of losses and dilution are minimal. Consequently, there was no need to incorporate provisions for ore loss or dilution in the finalized mine plan.

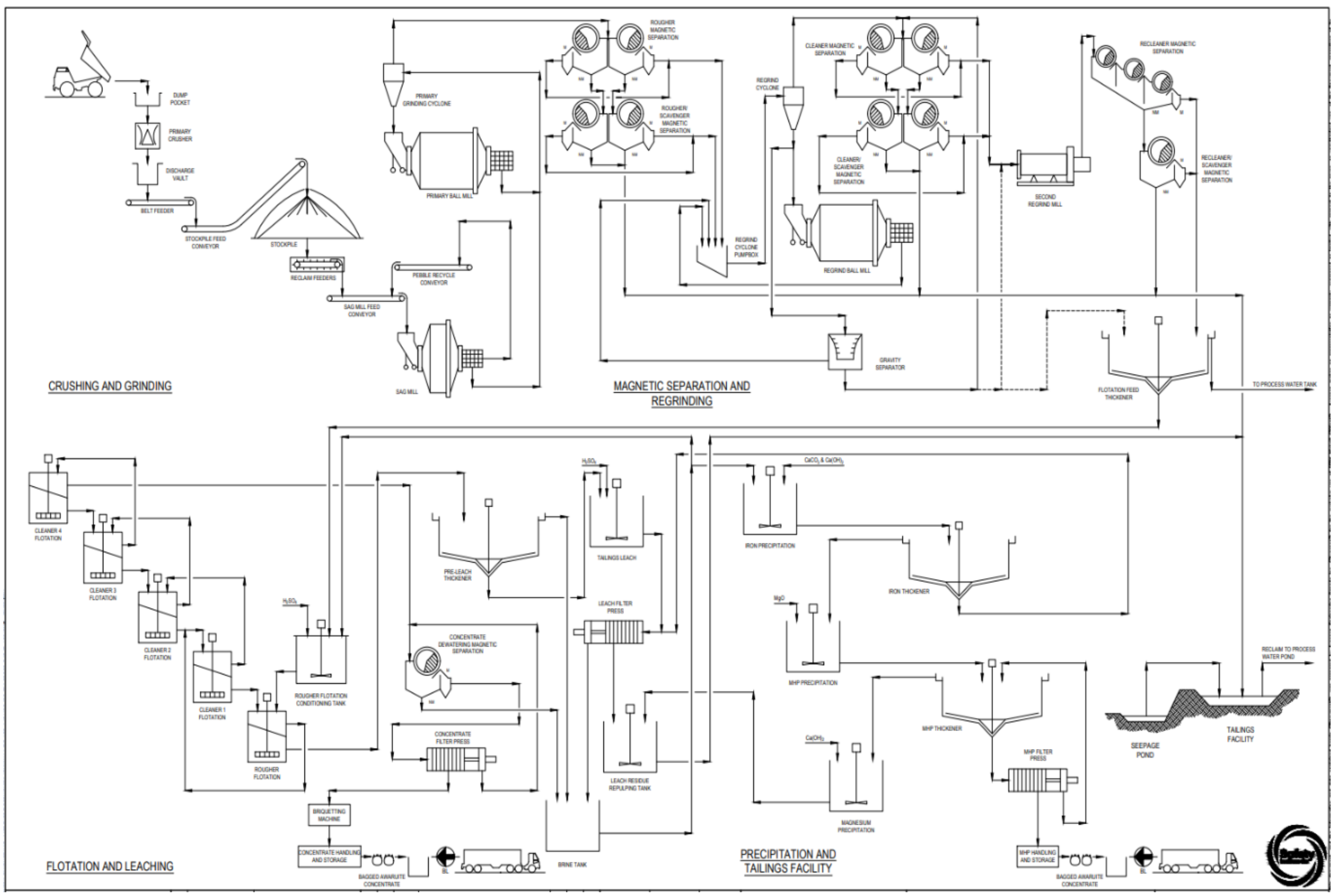
Costs are based on a "first principles" build-up of capital and operating costs for the life of the project with current vendor quotations for main consumables and maintenance. Capital costs for mining equipment were based on vendor budgetary quotations.

1.13 Recovery Methods

The process plant has been designed to recover nickel to produce a main product of an awaruite concentrate by magnetic and flotation treatment, and a MHP coproduct by precipitation of the nickel leached into solution during flotation and flotation tailings leach stages.

The selected process route is based on the metallurgical testwork performed since 2012 on samples from the Baptiste Deposit, as well as relevant industry benchmark data and Ausenco Process experience. The plant will be constructed in two phases. The initial plant capacity will be 108 kt/d ore feed, followed by an expansion for Year 10, which will be achieved by installing an additional operating line to increase the total processing capacity to 162 kt/d ore feed. A schematic of the process plant flowsheet is shown in Figure 1-4. Future stages of study will evaluate the opportunity to integrate the Phase 2 processes into the original concentrator facility, in whole or in part. Only the initial phase of the process plant is described in this section as the expansion follows the same flowsheet.

Figure 1-4: Overall Process Flow Diagram



Source: Ausenco, 2023.

Run-of-mine (ROM) ore material will be crushed in a gyratory crusher and conveyed to a covered stockpile from which the crushed ore will be reclaimed and conveyed to two parallel grinding lines, each including SAG milling followed by ball milling in closed circuit with cyclones (SAB). Overflow from the primary grinding cyclones will gravity feed to the magnetic separation circuit.

The magnetic separation circuit will consist of a three-stage separation process via low intensity magnetic separators (LIMS) and two regrind stages including:

- The magnetic concentrate from the rougher and rougher scavenger magnetic separation circuit will be reground in the first regrind circuit, which will include a ball mill in closed circuit with cyclones. A portion of the cyclone underflow will be treated in a centrifugal gravity concentrator to recover liberated nickel minerals. The gravity concentrate and first regrind cyclone overflow will be advanced to the cleaner magnetic separation circuit, while the gravity tailings will be sent back to the first regrind circuit.

- The magnetic concentrate from the cleaner and cleaner-scavenger magnetic separation circuit will be sent to the second regrind circuit, which will include a stirred mill configured in a direct-feed arrangement. The second regrind product will report to the recleaner magnetic separation circuit. The recleaner magnetic concentrate will be discharged to a thickener ahead of the awaruite flotation circuit.

- All the non-magnetic tailings will be combined and piped to the tailings facility.

The thickened magnetic concentrate will be further upgraded in the awaruite flotation circuit that will operate with mildly acidic conditions. The awaruite flotation circuit will include stages of conditioning, rougher flotation, and a closed 4-stage cleaner flotation circuit. Sulphuric acid will be dosed in the conditioning and cleaner flotation stages to maintain the required pH level.

The final awaruite concentrate will be dewatered via a LIMS followed by a filter press. The filtered awaruite concentrate will be compressed to produce awaruite concentrate briquettes. These briquettes are packaged into bulk bags and transported off-site via container trucks.

Nickel will also be recovered as a MHP coproduct from the nickel leached into solution in both the awaruite flotation and awaruite flotation tailings leach circuits. The awaruite flotation tailings will be treated via pre-leach thickening, acid leaching, and press filtration stages. The pre-leach thickener overflow and filter press filtrate will be combined to be reused in the awaruite flotation circuit. The excess brine will be advanced to a nickel recovery circuit which will include iron removal, MHP precipitation, and magnesium precipitation. The MHP precipitate will be thickened and filtered to produce a second saleable nickel product to be packaged and shipped off site. The magnesium precipitate will be sent to the tailings facility along with the filtered leach residue.

1.14 Project Infrastructure

1.14.1 Site Access

The current network of existing paved public roads and gravel forestry service roads (FSR) that connect from Fort St. James to the Baptiste site are 170 km in length. This distance will be reduced to 131 km with the construction of a new bridge across the Middle River, at a more southerly point than the existing crossing. Additional road upgrades to facilitate both the shorter route and four-season site access for mine traffic are planned for the construction phase and will include the following:

- a new 7 km resource road segment;
- a 253 m overall length steel girder with composite concrete deck bridge across Middle River;
- a 12 m overall length concrete slab girder bridge;
- a 24 m overall steel girder with composite concrete deck bridge; and
- four-season upgrades to 68 km of existing FSR segments.

There will also be a 16 km realignment of the existing Austin FSR that passes through the project site.

1.14.2 Site Infrastructure

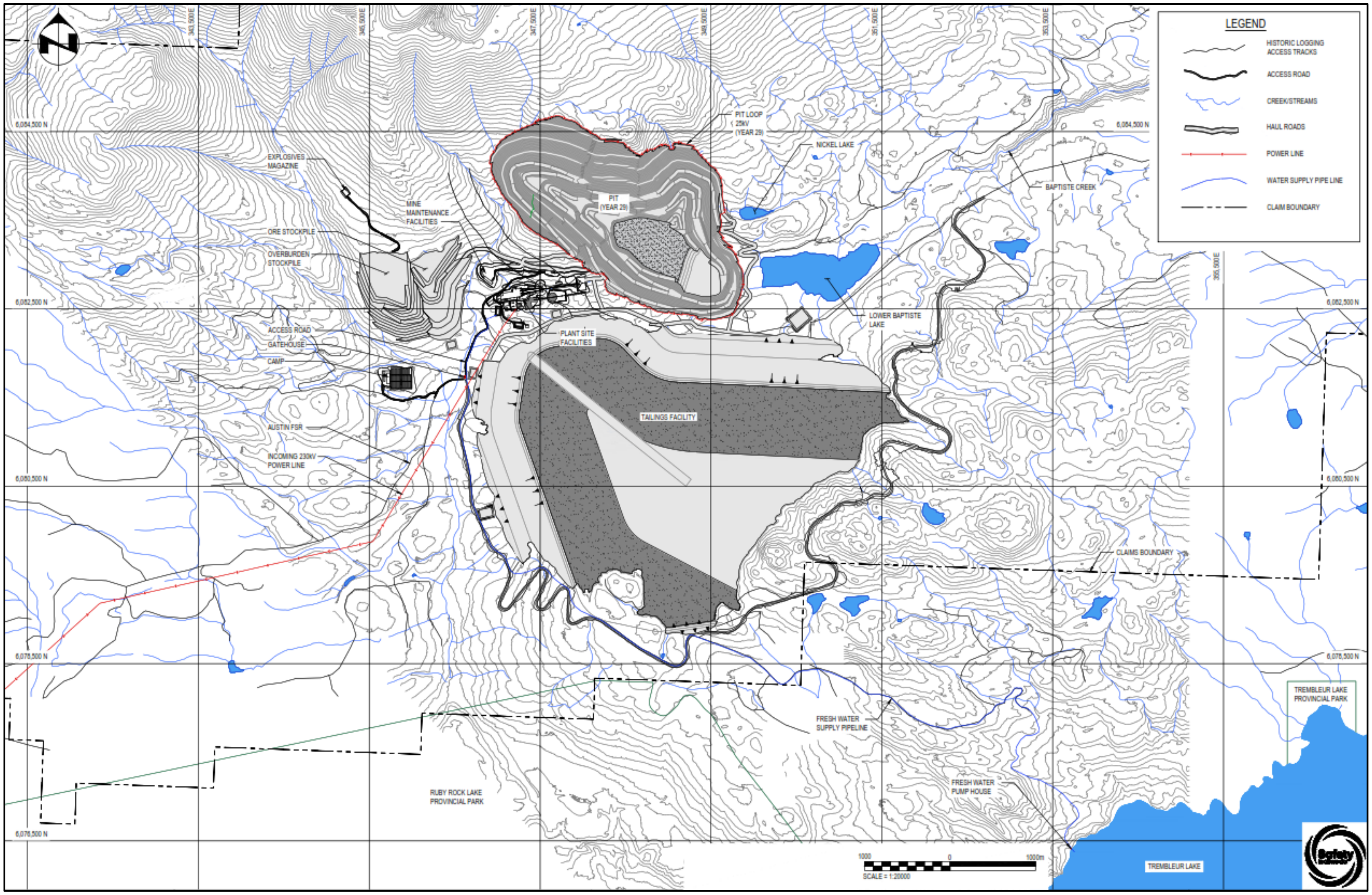
The process plant will be composed of several pre-engineered buildings that house the main processing circuits. The primary crusher will be located approximately 600 m east of the process plant.

The mine maintenance facilities are located within a dedicated pad adjacent to the process plant area. The Truck Shop includes four bays for mining and auxiliary equipment maintenance, with two additional bays added in Year 6. Other mine maintenance facilities include a truck wash and tire change facility, in addition to a mine dry, mine office, and mine mechanical shop.

The Project initially requires a 1,569-bed accommodation facility to support construction. The facility includes an initial 200-person early works camp followed by an additional 1,369 beds to be added progressively prior to the peak construction period. The permanent operations camp will utilize 325 of the 1,569 beds through the Phase 1 operation, with the remaining beds retained to support sustaining capital works, the Phase 2 expansion, and the increased requirements for Phase 2 operating personnel.

The overall site layout is shown in Figure 1-5.

Figure 1-5: Overall Site Layout



Source: Ausenco, 2023.

1.14.3 Site Power

The Project will require 145 MW of power in the first nine years of operations and an additional 74 MW for the remaining LOM operations to power the plant expansion to 162 kt/d. A 155 km, 230 kV overland transmission line, capable of transmission of 250 MW, will be constructed to connect to the BC Hydro grid at the Glenannan substation located 65 km west of Vanderhoof, BC. This point of interconnection has been assessed and found to be technically viable according to a formal study completed by BC Hydro in January 2023.

1.14.4 Tailings Facility

The proposed tailings facility is the key feature for the integrated waste and water management facilities that will be progressively developed and operated over the life of the mine. The PFS mine plan includes four waste materials to be managed within, or for use in construction of, the tailings facility, including: tailings, overburden from open pit stripping, Class A waste rock, and Class B waste rock. The tailings facility dams will be constructed to El. 1,120 masl, with a corresponding maximum dam height of approximately 190 m (measured from the crest of the dam to downstream toe) at an overall slope of 3H:1V. The tailings facility will store a total of approximately 1 Bm^3 of tailings, overburden, and Class B waste rock produced through open pit mining and mineral processing.

Tailings will be conveyed by gravity from the plant site through two DR11 high-density polyethylene (HDPE) pipelines (diameter ranging from 36” to 48”) to the tailings facility for discharge to Year 6 of operations. The total length of pipeline installed during pre-production (Year -1) is approximately 7 km. A tailings pumping system will be installed and operated starting in Year 7 to support ongoing tailing deposition as the height of the tailings facility rises to and above the elevation of the plant site. Additional tailings distribution pipelines will be added to facilitate discharge of tailings within the open pit in Year 28.

The conceptual site water management plan for mine operations includes collection of water in the tailings facility supernatant pond along with collection of contact water downstream of all other project infrastructure/disturbance and transfer to a process water pond at the plant site, where it will be reused in mineral processing. Water stored within the tailings facility pond will be reclaimed for use in mineral processing utilizing barge mounted vertical turbine pumps. Two barges each holding four pumps will be installed during the initial 108 kt/d phase of operations, with a third barge installed as the throughput increases to 162 kt/d, to manage the resulting increased water available for reclaim.

1.14.5 Freshwater Supply System

A range of potential water sources for the freshwater supply system were identified via a pre-screening assessment. Alternatives were selected by examining an approximately 20 km radius from the Project area and identifying major features which could potentially provide a freshwater source. Fresh water will be pumped from Trembleur Lake to a fresh water pond and gravity fed to the fresh water tank. The design of the freshwater pump and pipeline system includes a 14 km, 20” diameter, and covered on-grade steel pipeline at a constant pumping capacity year-round. The pumphouse will be constructed on the shore of the lake and a corridor comprising of a maintenance road, the pipeline and power line will connect the pumphouse to the processing plant.

1.15 Market Studies and Contracts

The PFS Base Case outlines the production of two saleable products from the Baptiste Nickel Project:

- Awaruite concentrate briquettes grading 60% nickel for sale as a feedstock to stainless steel production.
- MHP grading 45% nickel for sale as a feedstock to nickel sulphate production for the electric vehicle battery supply chain.

Over the life of the Project, it is anticipated that the awaruite concentrate will account for approximately 93% of the contained nickel output, with the remaining 7% of nickel output attributable to the MHP product.

Metallurgical testwork, as described in Section 13, has shown that the Baptiste Nickel Project can produce a clean, high-grade, awaruite concentrate through a mineral processing flowsheet using conventional unit operations. The concentrate will be dewatered, agglomerated into a briquette form, bagged, and placed into standard shipping containers for direct shipment to and use by stainless steel producers.

Based on the average projected analyst LME benchmark nickel price and the payability analysis presented in Table 1-8, Table 22-1 provides the basis for the selling prices assumed in the Economic Analysis presented in Section 22.

Table 1-8: Projected Analyst LME Benchmark Price

	Projected Price	
	US$ / lb Ni	US$ / t
Analyst 1	8.00	17,637
Analyst 2	8.00	17,637
Analyst 3	9.00	19,842
Analyst 4	9.00	19,842
Analyst 5	9.00	19,842
Analyst 6	9.50	20,944
Average	**8.75**	**19,290**

CRU, a leading provider of analysis and consulting in the mining, metals and fertilizer markets, prepared a market analysis report that looked at the ferronickel (FeNi) market and the comparability of the Baptiste awaruite concentrate briquettes to standard FeNi products. The analysis indicated that the high total percentage of nickel and metallic iron in the Baptiste awaruite product should provide an overall advantage for the product when compared to standard FeNi products. Additionally, the high nickel content of the Baptiste awaruite concentrate will reduce the amount of secondary elements charged into the melt and allow for the minimization or elimination of the more expensive Class 1 nickel typically used for trim in the stainless steel production process.

Overall, the Baptiste awaruite concentrate product should be able to substitute for standard FeNi in stainless steel production, and therefore attract similar payability to standard FeNi. Table 1-9 presents historical premium/discount data for standard FeNi, which averaged as a 5% discount to the LME nickel price over the last six calendar years. For the purposes of the economic analysis in Section 22 of this Report, payability of 95% of the LME nickel price has been assumed for the Baptiste awaruite concentrate product.

Table 1-9: Historical Premium / Discount for Standard FeNi

Year	Average LME Price (US$ /t)	Average Realized Price (US$ /t)	Premium / Discount (%)
2017	10,406	10,494	+1
2018	13,118	12,963	-1
2019	13,933	13,757	-1
2020	13,779	12,412	-10
2021	18,497	17,042	-8
2022	25,596	22,619	-12
Mean			**-5**
Median			**-4**

Source: Anglo American annual reports.

Based on publicly available market data, the payability for nickel content in MHP ranged from 70% to 90% of the LME nickel price over the 2020-22 period, with the low-end of that payability range coinciding with the period of extreme market volatility and elevated LME nickel prices in the first half of 2022. For the purposes of the economic analysis in Section 22 of this Report, payability of 87% of the LME nickel price has been assumed for the Baptiste MHP product.

Based on the average projected analyst LME benchmark nickel price and the payability analysis presented for the FeNi and MHP products, Table 1-10 provides the basis for the selling prices assumed in the Economic Analysis presented in Section 22.

Table 1-10: Selling Price for Nickel Products in PFS Economic Analysis

Price Reference	Awaruite Concentrate Product (US$/t Contained Ni)	MHP Product (US$/t Contained Ni)
LME price based on analyst average	19,290	19,290
Selling price used in PFS Economic Analysis	18,326	16,782

It should be noted that the cost of transport from the mine site to the destination port is included in the operating costs of the product; therefore, no adjustment to payability is necessary to account for product transportation. Transportation costs are discussed in Section 21 of this report.

The PFS Base Case economic analysis assumes that 100% of the Baptiste awaruite concentrate product will be sold directly to stainless steel producers over the entire life of the Project. In Section 24 of this report, the Refinery Option is presented whereby a large portion of Baptiste's total nickel production would be directed to the production of battery-grade nickel sulphate for the electric vehicle battery supply chain. A discussion on the marketability of Baptiste's battery-grade nickel sulphate in the Refinery Option is presented in Section 24.

In addition to the two nickel products, the Baptiste Nickel Project has the potential to upgrade flotation tailings to produce an iron-rich byproduct that can potentially be sold as an iron ore concentrate on a standalone basis. For this PFS, the sale of this iron ore byproduct (or any other byproduct(s)) is not considered in the economic analysis.

1.16 Environmental Studies, Permitting, and Engagement

1.16.1 Environmental Baseline Studies

The Baptiste Nickel Project has benefited from environmental baseline studies completed during two distinct periods: from 2011-2014, and 2021- present. The 2011-2014 environmental baseline studies were led by Cliffs Natural Resources, in parallel with project development activities at the time associated with the 2013 PEA. In 2014, Cliffs announced the intention to divest from the project, and baseline data collection ceased accordingly. In 2021, following issuance of the 2020 PEA, FPX began planning efforts to resume baseline study data collection. The current and in-progress baseline program was re-initiated during the summer of 2021 and is ongoing at the time of this Report.

Environmental baseline studies are planned to continue for the remainder of 2023 and into 2024. These baseline studies are in support of future environmental assessments and permitting for the Project, however the Project has yet to formally enter the environmental assessment process. At that time, it is anticipated that future direction will be obtained on the completion of all applicable baseline studies.

1.16.2 Environmental Assessment and Permitting

The environmental assessment and permitting framework for mining in Canada and British Columbia is well established and prescriptive with supporting guidance documents provided by government agencies. Currently, cultural and environmental baseline studies are ongoing to provide current conditions to inform the pending Provincial Environmental Assessment, Federal Impact Assessment, and permit applications. Individual permits to construct and operate the mine will only be approved for the project subsequent to the issuance of a provincial Environmental Assessment Certificate and the issuance of a federal Decision Statement.

1.16.2.1 Provincial Process

In BC, the Environmental Assessment Act (2018), defines whether a mine project is a reviewable project under the Act and Regulations if it exceeds certain thresholds. The Reviewable Projects Regulation (BC Reg. 67/2020), Part 3 – Mine Projects, Table 6 specifies that a new mine is a reviewable project if the annual production will exceed 75,000 t/a of mineral ore. This Project exceeds the threshold criteria and therefore is subject to review under the BC Environmental Assessment Act and Regulations. The Project must obtain an Environmental Assessment Certificate (EAC) prior to obtaining the required construction and operating permits. The BC Environmental Assessment Act review process uses a phased approach with imposed regulatory timelines. The Environmental Assessment (EA) is generally structured as follows:

- Identification and assessment of potential environmental, social, economic, cultural, and health impacts;
- Development of an acceptable scope and methodology for conducting the effects assessment of a selection of valued components (VCs);
- Characterization of residual effects potential for VCs after avoidance, mitigation measures, standard best management practices (BMPs), and monitoring programs are implemented;
- Prediction of the likelihood of significant residual effects occurring;

- Development of acceptable compensation measures to offset residual effects and maintain compliance with provincial and federal regulatory requirements as well as to effectively accommodate adversely affected Indigenous groups;
- Participation in Crown consultation proceedings to provide opportunities for Indigenous, federal, provincial, and local governments, stakeholders, special interest groups, and members of the public to learn about the Project, identify potential issues, provide input to potential avoidance/mitigation measures, and accommodate any infringement of Indigenous title and rights; and
- Incorporate economic, social, cultural, health, and environmental factors into proponent and government decision-making processes.

Key phases of the EA process in BC include:

- Early engagement phase (minimum of 90 days) occurring prior to submitting the detailed Project Description (s. 13 and s. 15 of the Act) and designed to attain consensus among participating Indigenous groups. It includes alternative dispute resolution options and leads to a Summary of Engagement and the Detailed Project Description.
- Remaining phases required to obtain an EA certificate (EAC) include:
  - EA Readiness phase and decision (s. 16(2), s. 17 or s. 18; 60 days minimum, but timeline is variable);
  - Process planning phase (120 days);
  - Application development and review phase (minimum of 180 days) and submission of final application;
  - Effects assessment and recommendation phases (150 days maximum); and
  - Decision phase (30 days maximum).

The Project will be bound by the conditions of the EAC. Post-certificate activities include mitigation effectiveness reports and may include audits, certificate amendments, extensions, and transfers.

1.16.2.2 Federal Process

The Government of Canada *Impact Assessment Act*, *2019* requires an Environmental Impact Assessment for federally designated projects. Projects that are not designated federally may still require screening in coordination with the provincial EA process. Government agencies responsible for coordinating the EA processes include the BC Environmental Assessment Office (EAO) and the Impact Assessment Agency of Canada (IAAC). The two agencies have put in place an agreement to address when a project falls under both provincial and federal environmental assessment responsibility, at which time the two governments will cooperate on the environmental assessment while retaining their respective decision-making powers.

The project will be assessed for whether it will require federal authorization by Fisheries and Oceans Canada (DFO) under paragraph 35(2)(b) of the Fisheries Act to commit harmful alteration, disruption, and destruction (HADD) of fish habitat or paragraph 34.4(2)(b) for any death of fish. An authorization would be required by the Minister, and habitat and/or productivity offsetting requirements acceptable to DFO and developed in consultation with participating Indigenous groups may need to be developed.

Whitebark pine and caribou are also valued components of federal legislative interest and will likely require specific studies and may require offset plans to be developed. Additional environmental baseline survey work, including the TEM and determination of the extent of fish habitat, wetlands, and presence of listed species and ecosystems will provide information for federal regulators to consider in the EA process.

1.16.2.3 Mine Permitting

The Baptiste Nickel Project is in the pre-development stage and has only been subject to Notice of Work permitting through the BC Ministry of Energy, Mines and Low Carbon Initiatives for seasonal surface exploration and geotechnical drilling programs. Anticipated permits required for the development and operation of the mine project include those listed in Table 20-3 Possible Provincial Authorizations, and Table 20-4, Possible Federal Authorizations. These permits will be pursued in parallel with the EA process.

1.16.3 Indigenous Engagement

The Baptiste Deposit is located within the traditional territories of Tl'azt'en Nation and Binche Whut'en, and First Nation traditional territories that overlap the wider Decar mineral claims area include Takla Nation, Lake Babine Nation, and Yekooche First Nation. Enduring relationships must be built with First Nations and Indigenous Rightsholders, based on transparency, accountability, mutual understanding, and respect for Indigenous rights and title. Since 2009, FPX has maintained regular engagement with First Nations on mineral exploration and project development activities, and meaningful and consistent engagement with Rightsholders will continue to be critical to decision-making processes throughout all aspects of the Project.

1.16.4 Mine Design and Closure Planning

The closure phase of the project will involve both active and long-term (post-closure) phases. Closure will be completed in a manner that will satisfy physical, chemical, and biological stability of the site. The site will be progressively reclaimed towards the end of operations and at the end of mining. The primary objective of the closure and reclamation activities will be to return the site to a self-sustaining condition, in a form that respects the current watershed function. Activities that will be carried out to achieve these objectives will include:

- Overburden will be re-handled from the overburden stockpile for placement on the downstream face of the tailings facility dam and the tailings beaches.
- A closure spillway will be constructed in the left abutment of the North Dam, to safely convey extreme storm event runoff.

Water from the tailings facility pond will be pumped to the open pit during active closure to accelerate development of an open pit lake. A spillway will be constructed at the south end of the open pit to safely convey extreme storm event runoff.

- A water treatment plant will be installed and will treat water from the open pit and/or tailings facility pond until such time that seasonal outflows meet water quality objectives for the receiving environment.
- The various water management ponds around site will only be removed when collected water meets the water quality objectives for the receiving environment.

- Mine site infrastructure (plant site, mining equipment, power line, etc.) will be removed and footprints of disturbed areas will be reclaimed.
- Instrumentation will be retained for use as long-term monitoring devices. Annual inspections of the site and ongoing evaluation of water quality, flow rates and instrumentation records will confirm design assumptions and performance of the closure activities.

FPX will continue to evaluate all aspects of the mine design, including alternatives for mine waste and water management and mine closure design, informed by future technical and environmental studies, and early engagement.

1.17 Capital and Operating Costs

1.17.1 Capital Cost Estimate

The initial, expansion, and sustaining capital cost estimates all conform to Class 4 guidelines for a PFS-level estimate with a ±25% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The closure capital cost estimate is an order-of-magnitude type. All capital cost estimates were developed in Q3 2023 US dollars (US$ or USD) based on budgetary quotations for equipment and construction contracts, as well as Ausenco's in-house database of projects and studies, which includes experience from similar operations.

The estimate includes mining, processing, on-site infrastructure, tailings and waste rock facilities, off-site infrastructure, project indirect costs, project delivery, owners' costs, and contingency. The total capital cost summary is presented in Table 1-11. The total initial capital cost for the Baptiste Nickel Project is US$2,182 M; the Year 10 expansion capital cost is US$763 M; and LOM sustaining capital cost is US$1,181 M. Closure capital costs are estimated at US$284 M.

Table 1-11: Summary of Capital Costs

WBS Level 1	WBS Description	Initial (US$M)	Expansion (US$M)	Sustaining (US$M)	Closure (US$M)	Total Cost (US$M)
1000	Mining	325	68	643	-	1,036
2000	Process Plant	730	379	0	-	1,109
3000	Tailings Facility	115	30	421	-	566
4000	On-Site Infrastructure	106	34	0	-	139
5000	Off-Site Infrastructure	127	1	0	-	128
	Subtotal Direct Costs	**1,403**	**511**	**1,064**	**-**	**2,978**
7000	Project Indirects	401	133	20	-	555
8000	Owner's Costs	106	16	0	-	122
9000	Contingency	272	103	97	-	471
Subtotal Indirect Costs		**779**	**252**	**117**	**-**	**1,148**
Project Total		**2,182**	**763**	**1,181**	**284**	**4,410**

Note: Values may not sum due to rounding.

1.17.2 Operating Cost Estimate

Table 1-12 and Table 1-13 summarize the overall operating cost of the project by operating phase. Table 1-14 summarizes the overall operating cost of the project over the LOM. The total operating cost for Phase 1 is estimated to average US$302 M per year, or US$7.88/t milled. For Phase 2, when the mine and concentrator throughput is increased to 162 kt/d, the annual total operating cost increases to an average of US$484 M per year, or US$8.24/t milled. On a LOM basis, operating costs average US$8.15/t milled.

Table 1-12: Operating Cost Summary – Phase 1 Average

Cost Area	Annual Total (US$Mt/a)	US$/t Milled	US$/lb Ni Produced	% of Total
Mining	99	2.59	0.98	32.9
Process	144	3.75	1.42	47.6
G&A	47	1.23	0.47	15.6
Product Transportation	12	0.31	0.12	3.9
Total	**302**	**7.88**	**2.99**	**100**

Note: Values may not sum due to rounding.

Table 1-13: Operating Cost Summary – Phase 2 Average

Cost Area	Annual Total (US$Mt/a)	US$/t Milled	US$/lb Ni Produced	% of Total
Mining	195	3.31	1.32	40.2
Process	211	3.59	1.43	43.6
G&A	62	1.05	0.42	12.7
Product Transportation	17	0.29	0.12	3.5
Total	**484**	**8.24**	**3.28**	**100**

Note: Values may not sum due to rounding.

Table 1-14: Operating Cost Summary – LOM Average

Cost Area	Annual Total (US$Mt/a)	US$/t Milled	US$/lb Ni	% of Total
Mining	164	3.14	1.24	38.5
Process	190	3.63	1.43	44.5
G&A	57	1.09	0.43	13.3
Product Transportation	15	0.29	0.12	3.6
Total	**427**	**8.15**	**3.21**	**100**

Note: Values may not sum due to rounding.

1.18 Economic Analysis

The economic analysis was performed assuming an 8% discount rate. The pre-tax net present value $NPV_{8\%}$ is US$2,923 M, the internal rate of return (IRR) is 19.1%, and payback is 4.6 years. On an after-tax basis, the $NPV_{8\%}$ is US$2,010 M, the IRR is 18.6%, and the payback period is 3.7 years. The after-tax payback period is shorter than the pre-tax payback period due to the inclusion of investment tax credit refunds in Year -1 to Year 2 of the Project.

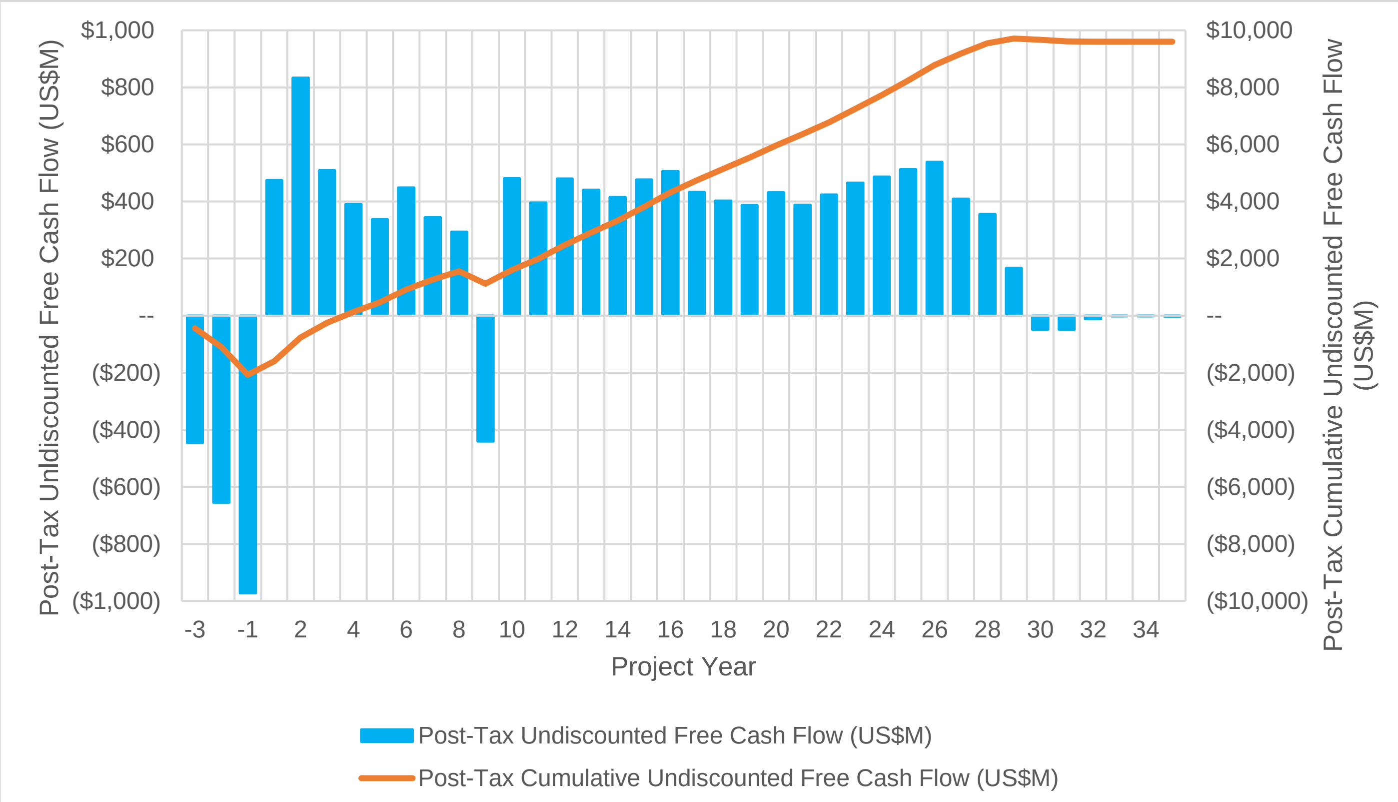
A summary of the project economics is included in Table 1-15 and shown graphically in Figure 1-6.

Table 1-15: Summary of Project LOM Cashflow Assumptions and Results

General	Unit	LOM Total / Average
Nickel Price	US$/lb	8.75
Exchange Rate	C$:US$	0.76
Discount Rate	%	8
Mine Life	years	29
Total Waste Tonnes Mined (excluding capitalized pre-stripping)	kt	837,576
Total Mill Feed Tonnes	kt	1,487,721
Strip Ratio (excluding capitalized pre-stripping)	waste:ore	0.56
Production	**Unit**	**LOM Total / Average**
Concentrator Head Grade, DTR Ni	%	0.13
DTR Ni Recovery – Awaruite Concentrate	%	82.2
DTR Ni Recovery – MHP	%	6.5
Total DTR Ni Recovery (Awaruite Concentrate + MHP)	%	88.7
Nickel Produced – Awaruite Concentrate	mlbs	3,505
Nickel Produced – MHP	mlbs	277
Total Nickel Produced (Awaruite Concentrate + MHP)	mlbs	3,781
Total Average Annual Nickel Production (LOM)	mlbs	130
Total Average Annual Nickel Production (LOM)	kt	59.1
Nickel Payability – Awaruite Concentrate	%	95
Nickel Payability – MHP	%	87
Total Payable Nickel	mlbs	3,570
Total Payable Nickel	Mt	1.62
Operating Costs	**Unit**	**LOM Total / Average**
Mining	US$/t milled	3.14
Processing	US$/t milled	3.63
General and Administration	US$/t milled	1.09
Concentrate Transport	US$/t milled	0.29
Total Operating	US$/t milled	8.15
Royalties	**Unit**	**LOM Total / Average**
NSR	%	1
C1 and AISC	**Unit**	**LOM Total / Average**
C1 Cost*	US$/lb Ni	3.70
All-in Sustaining Cost (AISC)**	US$/lb Ni	4.17
Capital Costs	**Unit**	**LOM Total / Average**
Initial	US$M	2,182
Sustaining	US$M	1,181
Growth	US$M	763
Closure	US$M	284
Total Capital Costs	US$M	4,410
Financials - Pre-Tax	**Unit**	**LOM Total / Average**
Pre-Tax $NPV_{8\%}$	US$M	2,923
Pre-Tax IRR	%	19.1
Pre-Tax Payback	years	4.6
Financials - Post-Tax	**Unit**	**LOM Total / Average**
Post-Tax $NPV_{8\%}$	US$M	2,010
Post-Tax IRR	%	18.6
Post-Tax Payback	years	3.7

* C1 costs consist of operating cash costs (which include transportation charges) plus treatment and refining charges.
** AISC consist of total cash costs (C1 costs) plus royalties, sustaining capital, BC hydro integration, and closure cost.

Figure 1-6: Projected Post-Tax Undiscounted Cash Flow



Source: Ausenco, 2023.

A sensitivity analysis was conducted on the Base Case pre-tax and post-tax $NPV_{8\%}$ and IRR of the Project, using the following variables: discount rate, nickel price, metallurgical recovery, operating costs, and initial capital costs. Post-tax sensitivity analysis results are summarized in Table 1-16. Analysis revealed that the project's NPV and IRR are most sensitive to changes in nickel price and metallurgical recovery, and due to the 29-year mine life, the project's NPV is more sensitive to operating costs than initial capital costs.

Table 1-16: Post-Tax Sensitivity Analysis Results

Post-Tax NPV (US$M) Sensitivity to Discount Rate

Discount Rate	Nickel Price				
	(20%)	(10%)	0%	10%	20%
5%	1,840	2,714	3,583	4,453	5,320
8%	837	1,427	2,010	2,593	3,173
10%	415	883	1,345	1,805	2,262

Post-Tax IRR Sensitivity to Discount Rate

Discount Rate	Nickel Price				
	(20%)	(10%)	0%	10%	20%
5%	13.0%	16.0%	18.6%	21.1%	23.4%
8%	13.0%	16.0%	18.6%	21.1%	23.4%
10%	13.0%	16.0%	18.6%	21.1%	23.4%

Post-Tax NPV (US$M) Sensitivity to Initial Capital Cost

Initial Capital	Nickel Price				
	(20%)	(10%)	0%	10%	20%
(20%)	1,051	1,635	2,217	2,796	3,375
(10%)	945	1,531	2,114	2,695	3,274
--	837	1,427	2,010	2,593	3,173
10%	728	1,321	1,907	2,490	3,071
20%	618	1,214	1,803	2,386	2,969

Post-Tax IRR Sensitivity to Initial Capital Cost

Initial Capital	Nickel Price				
	(20%)	(10%)	0%	10%	20%
(20%)	15.4%	18.8%	21.8%	24.6%	27.2%
(10%)	14.1%	17.2%	20.1%	22.7%	25.2%
--	13.0%	16.0%	18.6%	21.1%	23.4%
10%	12.0%	14.9%	17.4%	19.7%	21.9%
20%	11.2%	13.9%	16.3%	18.6%	20.6%

Post-Tax NPV (US$M) Sensitivity to Operating Cost

Operating Cost	Nickel Price				
	(20%)	(10%)	0%	10%	20%
(20%)	1,276	1,861	2,444	3,024	3,604
(10%)	1,057	1,644	2,227	2,809	3,388
--	837	1,427	2,010	2,593	3,173
10%	615	1,209	1,794	2,376	2,957
20%	392	989	1,577	2,160	2,742

Post-Tax IRR Sensitivity to Operating Cost

Operating Cost	Nickel Price				
	(20%)	(10%)	0%	10%	20%
(20%)	15.2%	17.9%	20.4%	22.8%	25.0%
(10%)	14.1%	17.0%	19.6%	22.0%	24.2%
--	13.0%	16.0%	18.6%	21.1%	23.4%
10%	11.7%	14.9%	17.7%	20.2%	22.6%
20%	10.5%	13.8%	16.7%	19.3%	21.8%

Post-Tax NPV (US$M) Sensitivity to Recovery

Recovery	Nickel Price				
	(20%)	(10%)	0%	10%	20%
(20%)	(135)	357	837	1,310	1,777
(10%)	357	896	1,427	1,952	2,476
--	837	1,427	2,010	2,593	3,173
10%	1,310	1,952	2,593	3,231	3,868
20%	N/A	N/A	N/A	N/A	N/A

Post-Tax IRR Sensitivity to Recovery

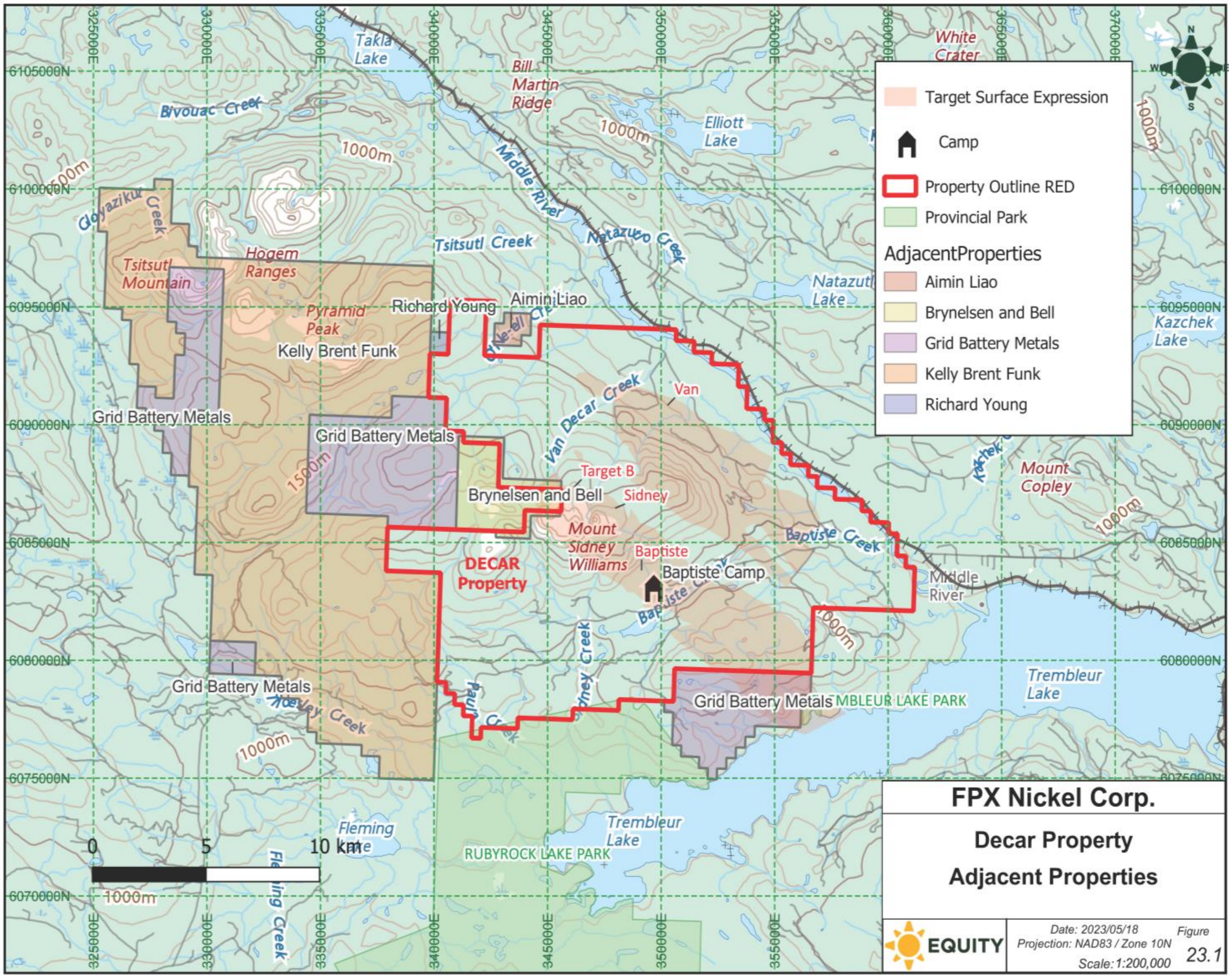
Recovery	Nickel Price				
	(20%)	(10%)	0%	10%	20%
(20%)	7.1%	10.2%	13.0%	15.4%	17.6%
(10%)	10.2%	13.3%	16.0%	18.4%	20.6%
--	13.0%	16.0%	18.6%	21.1%	23.4%
10%	15.4%	18.4%	21.1%	23.6%	26.0%
20%	N/A	N/A	N/A	N/A	N/A

Source: Ausenco, 2023.

1.19 Adjacent Properties

Mineral properties that lie adjacent to the Decar Property are held by Mr. Brynensen and Mr. Bell, Grid Battery Metals Inc. (Grid Battery), Mr. Kelly Brent Funk (MAC Property), Mr. Richard Young, and Mr. Aimin Liao. The location of these claim blocks relative to the Decar Property are shown in Figure 1-7 and three of these properties are described in more detail below. No information was found for the claims owned by Mr. Young and Mr. Liao.

Figure 1-7: Location of Adjacent Properties



Source: Equity Exploration, 2023.

The Mount Sidney Williams and MAC properties are located adjacent to and west of the Decar Property, though only the Mount Sidney Williams property has documented occurrences of awaruite that may be like the mineralization found at Decar. The MAC property, on the other hand, hosts a molybdenum-copper (Mo-Cu) porphyry deposit.

Information for the properties discussed in Section 23 has not been verified by any of the QPs responsible for this report and is not necessarily indicative of mineralization on the Decar Property.

1.20 Refinery Option

This PFS also presents an option for an off-site refinery (Refinery Option) to produce 40 kt/a of nickel in battery-grade nickel sulphate, along with cobalt and copper byproducts. The nickel sulphate and cobalt product will be marketed to the EV battery material supply chain, while the copper product will be marketed to the traditional copper supply chain. Any nickel in excess of the 40 kt/a in nickel sulphate will continue to be marketed directly to stainless steel producers. This Refinery Option is presented as a discrete option in Section 24 of this report. All other sections of this report pertain to the PFS Base Case exclusively.

The current assumption is a central BC refinery location, but the ultimate location of this refinery is dependent upon integrating the project into the downstream electric vehicle (EV) battery material supply chain, most likely as a component within an integrated battery hub. Inclusion of this off-site refinery would not require any material change to the design of the Baptiste site facilities, including the mine, concentrator, or infrastructure. The off-site refinery would process the majority of the awaruite concentrate and all of the MHP concentrate, with the remaining awaruite concentrate being shipped overseas for direct sale to stainless steel producers.

The refinery process consists of atmospheric leaching, pressure oxidation (POX) leaching, solvent extraction (SX), and crystallization and will produce three saleable products, including nickel sulphate, cobalt hydroxide, and a copper cementation product (copper cement). Nickel sulphate, cobalt hydroxide, and copper cement will be transported by rail to the market. A fourth product, sodium sulphate, is assumed to be free issued locally to kraft pulp mills for this study but could present an opportunity in the future.

Estimates of the overall circuit recovery have been made based on the presented testwork, industrial benchmarking and process modelling. The estimated recovery for payable metals in the refinery include 99.2% for nickel, 96.3% for cobalt, and 94.9% for copper.

The Refinery Option is designed to process awaruite and MHP concentrate produced by the Baptiste concentrator and will not influence mining. The Refinery Option has no effect on the concentrator.

The capital cost estimate conforms to Class 5 guidelines of AACE International, with an estimated accuracy of +50%/-35%. This conceptual study capital cost estimate was developed in Q3 2023 US$. Table 1-17 below summarizes the capital cost estimate and Table 1-18 summarizes the operating cost estimate for the Refinery Option.

Table 1-17: Capital Cost Estimate Summary – Refinery Only

Capital Category	Initial Capital (US$M)
Refinery	201
On-site Infrastructure	55
Off-site Infrastructure	5
Total Direct Costs	**262**
Project Indirects	73
Owner's Costs	13
Contingency	100
Total Capital Cost	**448**

Table 1-18: Operating Cost Estimate Summary – Refinery Only

Operating Cost Summary	Annual Total (US$M/a)	US$/t Ni
Power	5.6	139.4
Reagents and Consumables	64.8	1,618.2
Labour	12.9	321.5
Maintenance	2.9	71.4
Product Transportation	7.0	175.0
G&A	4.9	123.2
Total	**98.1**	**2,449**

Note: Values may not sum due to rounding.

Based on 2018 to 2022 market pricing data published by Morgan Stanley, the realized nickel sulphate premium and discount has been unstable, ranging from a significant discount (-40%) to a significant premium (+40%) to the LME nickel price. This is demonstrative of the currently small size and inherent volatility of the nickel sulphate market, which is further amplified by volatility of the overall nickel market during the same period, as demonstrated during the extreme spike in the LME price in early 2022. Based on market data published by Asian Metal, the 2022 nickel sulphate price in China ranged from a low of $23,677 to a high of $33,036/t of contained nickel.

As the nickel sulphate market grows in the coming years and preferred feedstocks are established, it is expected that a more consistent premium basis will be established based on typical upgrading costs. For the Baptiste nickel sulphate product, a premium of $1.00/lb Ni ($2,205/t Ni) to the assumed base LME nickel price of $8.75/lb ($19,290/t) has been applied in the Refinery Option economic analysis. This represents a realized price of $21,495/t of contained nickel, or an approximate 11% premium over the assumed base LME nickel price, or 17% more than the FeNi product in the PFS Base Case. The QP for this section has reviewed the studies and analyses and believes the results support the assumptions used in the economic analysis in Section 24.2.12.

Table 1-19 summarizes key criteria and results of the economic analysis for the Refinery Option.

Table 1-19: Summary of Project LOM Cashflow Assumptions and Results – Refinery Option

General	Unit	PFS Base Case + Refinery Option	Refinery Option Only***
		LOM Total / Average	
Nickel Price	US$/lb	8.75	8.75
Nickel Sulphate Premium (in addition to Nickel Price)	US$/lb	1.00	1.00
Cobalt Price	US$/lb	15.00	15.00
Cobalt Payability	%	87	87
Copper Price	US$/lb	3.50	3.50
Copper Payability	%	95	95
Exchange Rate	C$:US$	0.76	0.76
Discount Rate	%	8	8
Mine Life	years	29	29
Production	**Unit**	**LOM Total / Average**	
Total Average Annual Nickel Production	kt/a	58.8	39.1
Total Average Annual Cobalt Production	kt/a	0.6	0.6
Total Average Annual Copper Production	kt/a	0.3	0.3
Transportation and Royalties	**Unit**	**LOM Total / Average**	
NSR	%	1	0
C1 and AISC	**Unit**	**LOM Total / Average**	
C1 Cost*	US$/lb Ni	3.89	0.79
All-in Sustaining Cost (AISC)**	US$/lb Ni	4.38	0.79
Capital Costs	**Unit**	**LOM Total / Average**	
Initial	US$M	2,629	448
Sustaining	US$M	1,181	-
Growth	US$M	763	-
Closure	US$M	284	-
Financials - Pre-Tax	**Unit**	**LOM Total / Average**	
$NPV_{8\%}$	US$M	3,046	78
IRR	%	17.9	9.9
Payback	years	5.0	8.7
Financials - Post-Tax	**Unit**	**LOM Total / Average**	
$NPV_{8\%}$	US$M	2,127	63
IRR	%	17.7	9.9
Payback	years	3.9	7.5

* C1 costs consist of operating cash costs (including transport costs) plus treatment and refining charges for products excluding nickel sulphate.
** AISC consist of total cash costs plus royalties, sustaining capital, BC hydro integration, and closure cost.
*** Costs and financials for the Refinery Option Only are derived by simplistic delta cashflow analysis and not modeled as a standalone project.

1.21 Conclusions and Interpretations

The total indicated mineral reserves for the Baptiste Nickel Project are estimated at 1,488 Mt grading 0.13% DTR Ni and 0.21% Total Ni, for 1,933 kt of contained DTR Ni metal and 3,125 kt of contained Total Ni metal. Additionally, Inferred material totalling 44 Mt has been categorized as waste in the PFS.

Based on the assumptions and parameters presented in this report, the prefeasibility study shows positive economics (i.e. \$2,010 M post-tax $NPV_{8\%}$ and 18.6% post-tax IRR). The prefeasibility study supports a decision to advance the project to the feasibility stage of development.

The Refinery Option demonstrates the ability for the Baptiste Nickel Project to supply both the stainless steel and EV battery material supply chain. The economics of the Refinery Option are accretive, but more importantly demonstrate strategic flexibility.

1.22 Recommendations

Table 1-20 summarizes the proposed budget to advance the project through to FS completion.

Table 1-20: Proposed Feasibility Study Budget

Function	Cost (US$M)
Geology and Resource	5.4
Mining	0.4
Metallurgy	2.3
Process and Infrastructure	6.4
Waste and Water Management	4.5
Environment and Engagement	6.4
General	9.3
Total	**34.7**

1.22.1 Geology and Resource

To support further development of the Baptiste Resource Estimate, conducting an infill resource drilling program targeting the conversion of resources from the Inferred category to the Indicated category is recommended. A program of approximately 8,075 m will support conversion of the bulk of Inferred resources from the PFS resource model, which total approximately 44 Mt. The recommended resource drilling program will commence late in the second quarter of 2024 and complete early in the fourth quarter of 2024.

In addition to the above-described resource drilling, the following drilling will also help inform updates to the Baptiste Resource Estimate and general geological understanding of the overall claims package:

- Condemnation drilling totalling 1,000 m in the plant site, tailings facility, and key infrastructure areas.
- Open pit geotechnical and hydrogeological drilling program.

- Geotechnical and hydrogeological drilling in the overall project area.

Results from the proposed feasibility resource drilling program will collectively inform updates to the Baptiste geological model. The updated Baptiste geological model will then serve as the basis for an updated Baptiste Resource Estimate.

A re-logging program of pre-2017 drill core is also proposed for the Baptiste Deposit. This re-logging program would allow for detailed alteration logging of drill core to consolidate different vintages of logging; it would also allow for integration of the alteration coding into the geological model.

1.22.2 Mining

During the first half of 2024, is it recommended to complete several trade-off studies to define the opportunity to further increase project value, including:

- Decarbonization of pit haulage to reduce operating costs and carbon intensity, including evaluation of options such as trolley-assist, battery or hydrogen-powered haul trucks, and alternative fuel types.
- Evaluating autonomous haulage for potential to improve mine productivity and reducing on-site personnel requirements.
- Evaluation of semi-autonomous drills, reducing on-site personnel, eliminating exposure to potential hazards, and increase asset utilization as well as overall improved drilling and blasting performance (through increased accuracy).
- Optimization of mine planning parameters, including elements such as 400 t class haul trucks, 15 m bench heights, and a potential elevated cut-off grade in early mining years to increase mill feed grade.
- As part of the re-logging program recommended in the geology and resource section, the overburden thickness, boundaries and geomechanical characteristics can be re-evaluated. Note that the PFS adopted a conservative strategy, assuming that all the overburden material necessitates drilling and blasting.

In addition, conducting blasting simulations to refine the run-of-mine product size distribution assumption is recommended.

1.22.3 Metallurgy

The metallurgical work outlined below is recommended for the next project phase.

Execution of pilot testwork to further demonstrate the metallurgical flowsheet at higher throughput and longer duration, including:

- Piloting of approximately 60-80 tonnes of bulk sample material.
- Piloting of approximately 3-6 tonnes of a new master composite sample.
- Further expansion to the comminution database through variability testwork on individual drill hole intervals and drill hole composites.

Further testwork to optimize the tailings leach circuit design, including:

- Leach optimization testwork, specifically focused on leaching conditions and acid consumption.
- Testwork validating the purification and precipitation unit operations.
- Dewatering testwork, focusing on concentrate, MHP, and tailings leach residue.
- Briquetting testwork to confirm briquetting criteria and product quality.
- Sulphide flotation testwork to determine the potential opportunity for nickel sulphide recovery.
- Iron ore testwork to determine the potential opportunity to produce a magnetite iron ore concentrate from tailings leach circuit residue.

In addition to the above test work which will support the PFS Base Case, additional hydrometallurgical test work is recommended to support further advancement of the Refinery Option (see Section 24). Conducting a larger-scale test work program on the awaruite concentrate generated from the pilot scale test work program above is recommended. This test work program will focus on demonstrating the leaching process at semi-pilot scale and demonstrating of the purification and crystallization unit operations at a larger bench scale.

1.22.4 Process and Infrastructure

Completing several value engineering exercises is recommended for defining opportunity to further increase project value including:

- Trade-off study to evaluate acceleration of the metal production profile by means of executing the expansion to 162 kt/d earlier than Year 10, or a potential higher capacity initial build. This would include assessment of the effects on plant layout, camp facilities and infrastructure.
- Review and refinement of the grinding area of the concentrator, targeting an optimal blend of direct capital cost efficiency, constructability, operability and maintainability.
- Review and refinement of the overall construction execution approach to further refine the Project footprint (including temporary facility areas), execution approach, and overall execution duration.
- Trade-off study to evaluate the potential opportunity to produce a magnetite iron ore concentrate through upgrading of tails leach circuit residue.
- Trade-off study to evaluate a potential off-site administration center to support the Baptiste mine operations, to allow non-production roles to be located in a more urban setting.

The above value trade-off studies will be executed during the 2024 calendar year.

For the off-site power system, it is recommended design to advance to support formal entrance to the BC Hydro connection queue in 2024.

A formal logistics study is recommended for completion in 2024 to support the FS design basis, including identification of preferred shipping and other transit (road/rail) routes, maximum shipping envelopes, and maximum shipping weights.

In addition, select engineering and design elements will be advanced as required to support preparation of the Initial Project Description (IPD).

1.22.5 Waste and Water Management

The following is a list of recommendations for additional site investigations, testwork, and design studies to support FS design of the waste and water management facilities, primarily, the tailings facility:

- Continue early engagement with local communities and iterate tailings alternatives assessments accordingly with learnings from engagement and ongoing cultural and environmental baseline studies.
- Undertake detailed geotechnical and hydrogeological site investigation program(s) for the tailings facility dam foundations, water management pond foundations, other site infrastructure and open pit. A key requirement of the site investigation program(s) will be collection and laboratory testing of undisturbed samples of the tailings facility foundation materials.
- Complete detailed stability and seepage modelling of the proposed tailings facility dams based upon the results of the detailed site investigation program and laboratory testing of undisturbed samples.
- Complete a consolidation model to evaluate the change in density of the tailings mass throughout the mine life. The model will also provide an estimate of consolidation seepage (resulting from densification of the tailings mass) for use in an updated water balance model.
- Update the hydrometeorology inputs as further site-specific data is collected and develop a water balance model using a long-term synthetic climate and precipitation data set, to evaluate all years of mine operations and into closure.
- Optimize annual dam construction staging with overburden, Class A, and Class B waste rock release from the mine. Investigate the possibility of advancing mining of additional Class A waste rock to offset Class B waste rock presently considered for use in the first five years of dam construction.
- Review and update the dam construction concept and proposed construction materials with the results of ongoing geochemical testing programs.
- Complete tailings material testwork as required to support FS design.

1.22.6 Environment and Engagement

The following key activities are recommended to be undertaken to support ongoing project development and advancement to the next stage of engineering and entrance into the environmental assessment process:

- Continued early engagement with local communities and Indigenous Rightsholders.
- Cultural and environmental baseline data collection and reporting.
- Identification and initiation of additional environmental and engineering studies as required to support increased project definition and incorporate learnings from engagement and baseline programs.
- Formal entrance into the Environmental Assessment process with provincial and federal agencies.

1.22.7 Refinery Option

The following are recommendations for the conceptual level Refinery Option:

- Metallurgy
    - Conduct continuous testing on the counter-current leach to assess circuit performance under steady-state conditions.
    - Confirm solid-liquid separation properties of the POX and atmospheric leach residues.
    - Optimize Cu cementation and Al/Cr removal circuit to minimize the associated nickel loss to these products.
    - Evaluate the use of use of different neutralizing reagents, producing different salt products (for example using ammonia and forming ammonium sulphate) to reduce operating costs.
    - Conduct continuous CoSX testing and/or circuit modelling to determine the required number of extract and scrub stages to achieve the required Co over Ni selectivity and minimize nickel co-extraction.
    - Conduct continuous NiSX testing and/or circuit modelling to determine the required number of extract and scrub stages to achieve the required Ni over Mg selectivity and maximize nickel co-extraction and to determine the required number of strip stages to maximize strip extent while producing strip solution with acceptable acidity to feed nickel sulphate crystallization.
    - Conduct additional crystallization testing to define the required purge rates and maximum acceptable impurity and acidity in the mother liquor while maintaining acceptable product specifications.
    - Determine the quality of the salt byproduct (sodium sulphate or other depending on selected neutralizing reagent).
- General
    - Through advanced discussions with potential strategic partners, identify:
        - A specific location for the off-site refinery.
        - Process simplification options, such as elimination of the nickel sulphate crystallization circuit.
        - Best value waste disposal strategy.
        - Potential market for refinery byproducts.

2 INTRODUCTION

2.1 Introduction

FPX Nickel Corp. (FPX) commissioned Ausenco Engineering Canada Inc. (Ausenco) to compile a preliminary feasibility study (PFS) of the Baptiste Nickel Project (The Project). The PFS was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1.

The responsibilities of the engineering companies who were contracted by FPX to prepare this report are as follows:

- Ausenco managed and coordinated the development of this report, developed PFS-level designs for the process plant and general on-site infrastructure, developed the consolidated cost estimates, and prepared the economic analysis.
- Carisbrooke Consulting Inc. (Carisbrooke) provided the off-site power infrastructure design and costing.
- Equity Exploration Consultants Ltd. (Equity Exploration) completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, exploration works, sample preparation and analysis, and data verification.
- ERM Consultants Canada Ltd. (ERM) conducted a review of the environmental studies and permitting information.
- International Metallurgical and Environmental Inc. (IMI) coordinated and reviewed the metallurgical test results and defined process design criteria.
- Knight Piésold Ltd. (KP) supervised geotechnical investigations, provided geotechnical recommendations, and developed the PFS-level design and cost estimate of the tailings and water management facilities.
- Next Mine Consulting Ltd. (Next Mine) developed the mineral resource estimate.
- Onsite Engineering Ltd. (Onsite Engineering) prepared the design and cost estimate for off-site road access.
- TechSer Mining Consultants Ltd. (TechSer) designed the open pit mine, mine production schedule, and mine capital and operating costs.

2.2 Terms of Reference

The report supports disclosures by FPX in a news release dated September 6, 2023 titled, “FPX Nickel Delivers PFS for Baptiste Nickel Project with After-Tax NPV of US$2.01 Billion and 18.6% IRR.”

2.3 Qualified Persons

The qualified persons (QPs) for this technical report and the report sections for which each QP is responsible are listed in Table 2-1. By virtue of their education, experience, professional association, and independence from FPX, the individuals presented in Table 2-1 are each considered to be a QP as defined by NI 43-101.

Table 2-1: Report Contributors

Qualified Person	Professional Designation	Position	Employer	Independent of FPX	Report Section
Kevin Murray	P.Eng.	Manager, Process Engineering	Ausenco Engineering Canada Inc.	Yes	1.1, 1.13, 1.14.2, 1.14.5, 1.15, 1.17, 1.18, 1.20, 1.21, 1.22.4, 1.22.7, 2, 3.1, 3.3,3.4, 17, 18.1, 18.2.3-18.2.5, 18.3-18.6, 18.9.2, 18.10, 18.11.2,18.11.3, 18.12,18.13, 19, 21.1, 21.2.1, 21.2.2, 21.2.3.1, 21.2.4, 21.2.6, 21.2.7.2, 21.2.8, 21.2.9, 21.3.1, 21.3.2, 21.3.4-21.3.8, 22, 24.1, 24.2.1-24.2.9, 24.2.11,24.2.12, 25.1, 25.8, 25.10-25.13, 25.14.1.3, 25.14.1.6.2-25.14.1.6.3, 25.14.1.7, 25.14.1.9, 25.14.2.3, 25.14.2.7, 26.1, 26.2.4, 26.2.7, 26.3, 27
Ronald Jaap Voordouw	P.Geo.	Partner, Director Geoscience	Equity Exploration Consultants Ltd.	Yes	1.2-1.8, 1.19, 1.22.1, 4-12, 23, 25.2-25.4, 25.14.1.1, 25.14.2.1, 26.2.1, 27
Jeffrey B. Austin	P.Eng.	President	International Metallurgical and Environmental Inc.	Yes	1.9, 1.22.3, 13, 25.5, 25.14.1.2, 25.14.2.2, 26.2.3, 27
Cristian H. Garcia Jimenez	P.Eng.	Manager, Strategic Mine Planning	TechSer Mining Consultants Ltd.	Yes	1.11, 1.12, 1.22.2, 15, 16, 18.2.2, 21.2.3.2, 21.3.3, 25.7, 25.14.1.5, 25.14.2.4, 26.2.2, 27
Duke Barret Reimer	P.Eng.	Civil Engineer	Knight Piésold Ltd.	Yes	1.14.4, 1.22.5, 18.7, 18.8, 18.11.1, 21.2.5, 25.14.1.6.1, 25.14.2.5, 26.2.5, 27
Richard FJ Flynn	P.Geo.	Independent Mining Consultant, QP	Next Mine Consulting Ltd.	Yes	1.10, 14, 25.6, 25.14.1.4, 27
David John Baldwin	P.Eng.	President	Carisbrooke Consulting Inc.	Yes	1.14.3, 18.9.1, 21.2.7.3, 27
Paul Mysak	P.Eng.	Supervising Engineer, Principal	Onsite Engineering Ltd.	Yes	1.14.1, 18.2.1, 21.2.7.1, 27
Harold Rolf Schmitt	P.Geo	Technical Director, Permitting	ERM Consultants Canada Ltd.	Yes	1.16, 1.22.6, 3.2, 20, 24.2.10, 25.9, 25.14.1.8, 25.14.2.6, 26.2.6, 27

2.4 Site Visits and Scope of Personal Inspection

A summary of the site visits completed by the QPs is presented in Table 2-2.

Table 2-2: Site Visits

Qualified Person	Date of Site Visit	Days on Site
Paul Mysak, P.Eng.	November 25-26, 2021	2
Ronald Jaap Voordouw, P.Geo.	July 4-7, 2022	4
Richard FJ Flynn, P.Geo.	July 4-7, 2022	4
Cristian H. Garcia Jimenez, P.Eng.	June 27, 2023	1
Kevin Murray, P.Eng.	June 27, 2023	1
Duke Reimer, P.Eng.	June 27, 2023	1

Paul Mysak visited the visited the site between November 25, 2021, and November 26, 2021. Activities during the site visit included the following:

- Review of the existing FSR network, including roads and stream crossings.
- Review of potential sites for the proposed Middle River Bridge.

Ronald Jaap Voordouw and Richard FJ Flynn visited the site between July 4, 2022 and July 7, 2022. Activities during the site visit included the following:

- Observing the outcrops at the northwestern end of the Baptiste Deposit.
- Reviewing FPX's data and management of the 2017 and 2021 drill programs.

Kevin Murray, Cristian H. Garcia Jimenez, and Duke Reimer visited the site June 27, 2023. Activities during the site visit included the following:

- Reviewing of the general topography and proposed location of site facilities.
- Review of the current access road.
- Reviewing of the locations for proposed key facilities, including the open pit mine, concentrator, infrastructure, and tailings facility.

2.5 Effective Dates

This technical report has three significant dates, as follows:

- Baptiste Nickel Project mineral resource estimate: November 14, 2022.
- Baptiste Nickel Project mineral reserve estimate: September 6, 2023.
- Financial analysis: September 6, 2023.

The effective date of this report is based on the date of the financial analysis, which is September 6, 2023.

2.6 Information Sources and References

This technical report is based on internal company reports, maps, published government reports, and public information as listed in Section 27. It is also based on information cited in Section 3.

The authors are not experts with respect to legal, socio-economic, land title, or political issues, and are therefore not qualified to comment on issues related to the status of permitting, legal agreements, and royalties. Information related to these matters has been provided directly by FPX and includes, without limitation, validity of mineral tenure, status of environmental and other liabilities, and permitting to allow completion of environmental assessment work. These matters were not independently verified by the QPs but appear to be reasonable representations that are suitable for inclusion in Section 4 of this report.

2.7 Previous Technical Reports

The Baptiste Nickel Project has been the subject of previous technical reports, as summarized in Table 2-3.

Table 2-3: Summary of Previous Technical Reports

Reference	Company	Technical Report
Amerlin Exploration Services Ltd., 2011	First Point Minerals Corp.	Report on the Decar Nickel Property
Caracle Creek International Consulting Inc., 2012	Cliffs Natural Resources Exploration Canada Inc. / First Point Minerals Corp.	Independent Technical Report – Decar Nickel Property, British Columbia, Canada
Caracle Creek International Consulting Inc., 2013	Cliffs Natural Resources Exploration Canada Inc. / First Point Minerals Corp.	Independent Technical Report – Resource Update – Decar Nickel Property, British Columbia, Canada
Tetra Tech, 2013	Cliffs Natural Resources Exploration Canada Inc. / First Point Minerals Corp.	Preliminary Economic Assessment – Decar Nickel Project, British Columbia, Canada
Equity Exploration Consultants Ltd., 2018	FPX Nickel Corp.	2018 Technical (N.I. 43-101) Report on the Decar Nickel-Iron Alloy Property
BBA Inc., 2020	FPX Nickel Corp.	Preliminary Economic Assessment – Baptiste Nickel Project

2.8 Currency, Units, Abbreviations and Definitions

All units of measurement in this report are metric and all currencies are expressed in US$ unless otherwise stated. A list of report abbreviations is provided in Table 2-4.

Table 2-4: List of Abbreviations

Abbreviation	Definition
%	Percent
°	Azimuth/ dip in degrees
°C	Degree Celsius
°F	Degree Fahrenheit
µg	Microgram
µm	Micron
2D	2-Dimensional
3D	3-Dimensional
a	Annum
(a)	Aqueous
AACE	Association for the Advancement of Cost Engineering
ACME	Acme Analytical Laboratories
Ai	Abrasion Index
AIR	Application Information Requirements
AISC	All-In Sustaining Cost
Al	Aluminum
ALARP	As Low As Reasonably Possible
AOP	Areas of Potential
As	Arsenic
ATS	Automatic Transfer Switch
ATV	Acoustic Televiewer
Au	Gold
AWi	Autogenous Work Index
BC	British Columbia
BMP	Best Management Practice
BWi	Bond Ball Mill Work Index
C	Carbon
C$	Canadian Dollar
C1	Dry Commissioning
C2	Wet Commissioning
C3	Ore Commissioning
Ca	Calcium
$CaCO_3$	Calcium Carbonate (Limestone)
CAD	Canadian Dollar
CaO	Calcium Oxide
CAPEX	Capital Expenditure
Cd	Cadmium

Abbreviation	Definition
CEA	Cumulative Expenditure Account
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	Centimeter
CM	Construction Management
CN	Canadian National
Co	Cobalt
CoSX	Cobalt Solvent Extraction
Cr	Chromium
CRM	Certified Reference Materials
CTMITC	Clean Technology Manufacturing Investment Tax Credit
Cu	Copper
CVavg	Coefficient of Variance
CWi	Crusher Work Index
DCIP	Downhole Induced Polarization and Resistivity
DDH	Diamond Drill Hole
DFO	Department of Fisheries and Oceans Canada
DGI	DGI Geoscience Inc.
DGPS	Differential Global Positioning System
DL	Detection Limit
DTR	Davis Tube Recoverable
EA	Environmental Assessment
EAC	Environmental Assessment Certificate
EAO	Environmental Assessment Office
EEM	Environmental Effects Monitoring
EEP	Engineering Execution Plan
EGL	Effective Grinding Length
EM	Electromagnetic
EOH	End of Hole
EPCM	Engineering, Procurement and Construction Management
ERM	ERM Consultants Canada Ltd.
EUR	European Dollar
EV	Electric Vehicle
F_{80}	80% passing feed size
Fe	Iron
Fe_2O_3	Iron (III) Oxide
FEED	Front-End Engineering and Design
FeNi	Ferronickel
FIFO	Fly-in-fly-out

Abbreviation	Definition
FPX	FPX Nickel Corp.
FS	Feasibility Study
FSR	Forestry Service Road
(g)	Gaseous
g	Gram(s)
G	Gauss
G&A	General and Administrative
g/cm^3	Gram per Cubic Centimeter
GM	Geological Model
GMD	Gearless Mill Drive
GPS	Global Positioning System
GSC	Geological Survey of Canada
h	Hour
H	Hydrogen
H_2SO_4	Sulphuric acid
Ha	Hectares
HADD	Harmful Alteration, Disruption, and Destruction
HDPE	High-Density Polyethylene
HPAL	High Pressure Acid Leach
HPGR	High Pressure Grinding Rolls
HSEC	Health, Safety, Environment, and Community
HV	High Voltage
Hz	Hertz
IAAC	Impact Assessment Agency of Canada
IBC	Intermediate Bulk Containers
ICP-OES	Inductively Coupled Plasma Optical Emission Spectrometry
ID2	Inverse Distance Squared
ILS	Intermediate Leach Solution
IMI	International Metallurgical & Environmental Inc.
INS	Inertial Navigation System
IP	Induced Polarization
IPD	Initial Project Description
IRR	Internal Rate of Return
JKDW	JK Drop Weight
K	Potassium
KCB	Klohn Crippen Berger
kHz	Kilohertz
km	Kilometer

Abbreviation	Definition
km^2	Square Kilometre
KP	Knight Piésold Ltd.
kt	Kiltotonnes (1,000 tonnes)
kt/a	Kiltotonnes per annum (1,000 tonnes per annum)
kt/d	Kilotonnes per Day (1000 tonnes per day)
kV	Kilovolts
kW	Kilowatt
kWh/t	Kilowatt hour per tonne
L	Liters
lb	Pound
LG	Lerchs Grossmann
LIDAR	Light Detection and Ranging
LIMS	Low Intensity Magnetic Separator
LME	London Metal Exchange
LMMP	Logistics and Materials Management Plan
LOI	Loss on Ignition
LOM	Life of Mine
LRS	Liquid Resistance Starter
m	Meter
M	Mega (Million)
m^2	Square meter
m^3	Cubic meter
m^3/h	Cubic meter per hour
masl	Meters Above Sea Level
MCC	Motor Control Center
Mg	Magnesium
MgO	Magnesium Oxide
MHP	Mixed Hydroxide Precipitate
min	Minute
mm	Millimetre
Mn	Manganese
MoA	Memorandum of Agreement
Mt	Million tonnes
Mt/a	Million tonne per annum
MTO	Mineral Titles Online
mV/V	Millivolt per Volt
MVA	Megavolt Amperes
MVR	Mechanical Vapor Recompression

Abbreviation	Definition
Na	Sodium
NAD	North American Datum
NaOH	Sodium Hydroxide
NCS	Non-Certified Standards
Ni	Nickel
Ni_3Fe	Awaruite
NiSX	Nickel Solvent Extraction
NPI	Nickel Pig Iron
NPV	Net Present Value
NSR	Net Smelter Return
NTS	National Topographic System
NW	Northwest
OK	Ordinary Kriging
OPEX	Operating Expenditure
OTV	Optical Televiewer
P	Phosphorus
P_{80}	80% passing product size
Pb	Lead
PCP	Project Controls Execution Plan
PEA	Preliminary Economic Assessment
PEP	Project Execution Plan
PFS	Preliminary Feasibility Study
PGA	Peak Ground Acceleration
PGDC	Prince George Distribution Center
PGE	Platinum Group Elements
pH	Potential of hydrogen
PLS	Pregnant Leach Solution
POX	Pressure Oxidation
ppb	Parts per Billion
ppm	Parts per Million
QA/QC	Quality Assurance/ Quality Control
QEMSCAN	Quantitative Evaluation of Materials by Scanning Electron Microscopy
QP	Qualified Person
RKEF	Rotary Kiln - Electric Furnace
ROM	Run-of-mine
ROW	Right-of-Way
RQD	Rock Quality Designation
RTK	Real Time Kinematic

Abbreviation	Definition
RWi	Bond Rod Mill Work Index
(s)	Solid
s.	Section
S	Sulphur
SAB	SAG and Ball Mill Circuit
SAF	Submerged Arc Furnace
SAG	Semi Autogenous Grinding
SD	Standard Deviation
SE	Southeast
sec	Second
SEDAR	System for Electronic Document Analysis and Retrieval
SELP	Sasuchan Environmental LP
SG	Specific Gravity
Si	Silicon
SIP	Species Identification Protocol
SIPX	Sodium Isopropyl Xanthate
SMC	Steve Morrell Comminution
SR W/O	Ratio between Waste and Ore
SX	Solvent Exchange
t	Metric Tonne(s)
TEM	Terrestrial Ecosystem Mapping
TMI	Total Magnetic Intensity
TOH	Top of Hole
t/a	Tonnes per Annum
t/d	Tonnes per Day
t/h	Tonnes per Hour
TSS	Total Suspended Solids
UN	United Nation
US$	United States Dollar
USD	United States Dollar
UTM	Universal Transverse Mercator
V	Volts
VC	Valued Component
VLF-EM	Very Low Frequency Electromagnetic
VMS	Volcanogenic Massive Sulphide
VSA	Vacuum Swing Adsorption
W	Watt
w/w	weight by weight

Abbreviation	Definition
WAAS	Wide Area Augmentation System
WBS	Work Breakdown Structure
WRIM	Wound Rotor Induction Motor
wt%	Percentage by Weight
XRF	X-Ray Fluorescence
YXS	Prince George Airport
Zn	Zinc

3 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The QPs have relied on other expert reports which provided information regarding mineral rights, surface rights, property agreements, royalties, permitting, social and community impacts, taxation, and marketing for sections of this report.

3.2 Environmental, Permitting, Closure, and Social and Community Impacts

The QPs have fully relied upon, and disclaim responsibility for, information derived from FPX and experts retained by FPX for information related to permitting, and social and community impacts through the following:

- Allnorth Consultants Ltd., (2013a). Decar Nickel Property EBS Program Interim Quarterly Report September – November 2012. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.
- Allnorth Consultants Ltd., (2013b). 2012 Meteorology Baseline Study Report. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.
- Allnorth Consultants Ltd., (2013c). 2012 Hydrology Baseline Study Report. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.
- Allnorth Consultants Ltd., (2013d). 2012 Surface Water Quality Baseline Study Report. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.
- Allnorth Consultants Ltd., (2013e). 2012 Hydrogeology Baseline Study Report.
- Allnorth Consultants Ltd., (2013f). 2012 Vegetation Baseline Study Report. Report prepared by Allnorth for Caracle Creek International Consulting Inc. and Cliffs Natural Resources.
- Allnorth Consultants Ltd., (2013g). Soils Baseline Study Report. Report prepared by Allnorth for Caracle Creek International Consulting Inc. and Cliffs Natural Resources.
- Allnorth Consultants Ltd., (2013h). 2012 Fish and Fish Habitat Baseline Study Report. Report prepared by Allnorth for Caracle Creek International Consulting Inc. and Cliffs Natural Resources.
- Allnorth Consultants Ltd., (2013i). Aquatic Environment Baseline Study Report. Report prepared by Allnorth for Caracle Creek International Consulting Inc. and Cliffs Natural Resources.
- Allnorth Consultants Ltd., (2013j). Summary of 2012 Archaeological Assessments. Report prepared by Allnorth for Caracle Creek International Consulting Inc. and Cliffs Natural Resources.
- Allnorth Consultants Ltd., (2014a). 2012-2013 Terrestrial Wildlife Baseline Study Report. Report prepared by Allnorth for Caracle Creek International Consulting Inc. and Cliffs Natural Resources.
- Allnorth Consultants Ltd., (2014b). 2013-2014 Meteorology Baseline Study Report. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.

- Allnorth Consultants Ltd., (2014c). 2013-2014 Surface Water Quality Baseline Study Report. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.
- Allnorth Consultants Ltd., (2014d). 2013-2014 Hydrogeology Baseline Study Report. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.
- British Columbia Ministry of Environment (BCMoE), (2016). Water and Air Baseline Monitoring Guidance Document for Mine Proponents and Operators-Version 2.
- Caracle Creek International Consulting Inc., (2012). ARD and ML Assessment Report.
- Ecofor, (2022). Preliminary Field Reconnaissance of FPX Nickel's Decar Property Footprint.
- Klohn Crippen Berger (KCB), (2012). Decar Nickel Project – Preliminary Environmental Baseline Report.
- Shas-Ti Environmental, (2023a). 2022 Decar Aquatics Summary Report.
- Shas-Ti Environmental, (2023b). 2022 Decar Fisheries Summary Report.
- Shas-Ti Environmental, (2023c). 2022 Decar Vegetation Data Summary Report.
- Shas-Ti Environmental, (2023d). 2022 Decar Wildlife Data Summary Report.

This information is used in support of Sections 1.16, 16, 18, 20 and 25.9.

3.3 Taxation

The QPs have fully relied upon, and disclaim responsibility for, information supplied by FPX's third-party tax expert related to taxation calculations and assumptions as applied to the financial model, as received by email from FPX on August 18, 2023.

This information is used in Sections 1.18, 22 and 25.12.

3.4 Marketing

The QPs have fully relied upon, and disclaim responsibility for, information derived from FPX and experts retained by FPX for information on markets, including the following:

- CRU International Ltd report, July 31, 2023 titled "Baptiste PFS Support"
- FPX Nickel email, August 30, 2023, subject "FW: Baptiste – Marketing Chapter and Final Marketing Assumptions"

This information is used in Sections 1.15, 1.18, 19, 22 and 25.12.

4 PROPERTY DESCRIPTION AND LOCATION

The Decar Property comprises 62 contiguous mineral claims that cover 24,740 hectares (247 km^2) in the Omineca Mining Division of central BC, Canada (Figure 4-1, Figure 4-2). The approximate centre of the Property is at 54˚54'30.5" north latitude and 125˚21'31" west longitude. The grid system used by FPX for work on the Property is North American Datum of 1983, Universal Transverse Mercator zone 10 north (NAD83/UTM zone 10N), which defines the centre of the Property at 6,087,000 m north and 350,000 m east. The relevant National Topographic System (NTS) map sheets for the Property are 93K083, 084, 085, 093, 094 and 095.

Mineral Titles Online (MTO) is a mineral claim registry maintained by the Government of BC. MTO claim boundaries are defined by latitude and longitude so that they form a seamless grid without overlap. "Legacy" mineral claims were staked on the ground prior to the introduction of the MTO system and take precedence over MTO claims.

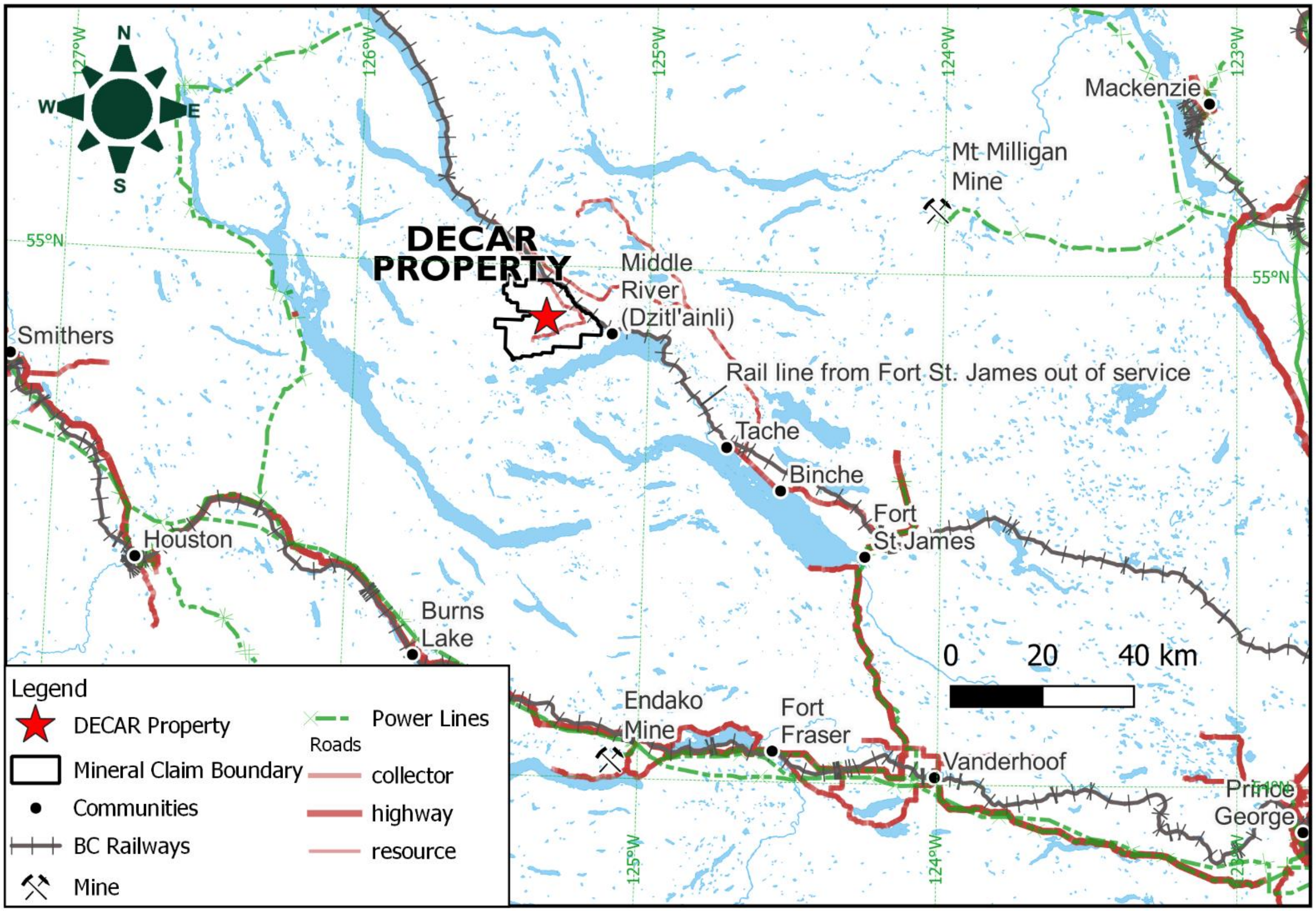
The claims confer title to subsurface mineral tenure only, as defined by the Mineral Tenure Act of BC. Surface rights over non-overlapping MTO claims are held by the Crown, as administered by the Government of BC. The ownership of other rights (placer, timber, water, grazing, trapping, outfitting, etc.) affecting the property was not investigated by the authors.

MTO claim data for the Decar Property is summarized in Appendix A and shown in Figure 4-2. Five claims along the southern boundary of the Property overlap with Rubyrock Lake Provincial Park and are off limits to exploration. Another 10 claims along the northeastern margin of the property overlap with Mineral Reserve Site 326751 where exploration is either restricted or not permitted. Together these overlaps cover 823.9 ha and so reduce the size of the property to 23,917 ha. All known nickel deposits and targets on the Property are located at least 2.5 km from the Mineral Reserve and 5 km from the Provincial Park boundary.

Another seven claims overlap with legacy claims and 14 claims overlap with District Lots. Mineral rights for those claims that overlap with legacy claims cover 270.0 ha and are held 50% each by Mr. Stuart Dal Brynelson and Mr. John Malcolm Bell. District Lots are surveyed land parcels that may be sold by the Crown to convey title (ownership) of the surface rights to the purchaser. Exploration on those claims overlapping District Lots is permitted but requires notification of the surface rights holder. The Van target lies within 1 km of District Lots 2036 and 2047 whereas the Baptiste Deposit, Sid target and B target are all located more than 4.5 km from District Lot boundaries.

British Columbia law requires assessment expenditures to maintain tenure ownership past the expiry dates for the mineral claims. As of the effective date of this report, annual assessment work requirements were set at C$5 per hectare (ha) for anniversary Years 1 and 2, C$10/ha for Years 3 and 4, C$15/ha for Years 5 and 6, and C$20/hectare for all subsequent years. Exploration expenditure can be distributed over contiguous claims for up to 10 years into the future. As of August 2023, all 62 of the Decar claims are in good standing until at least July 2032 (see Appendix A).

Figure 4-1: Location Map for the Decar Property



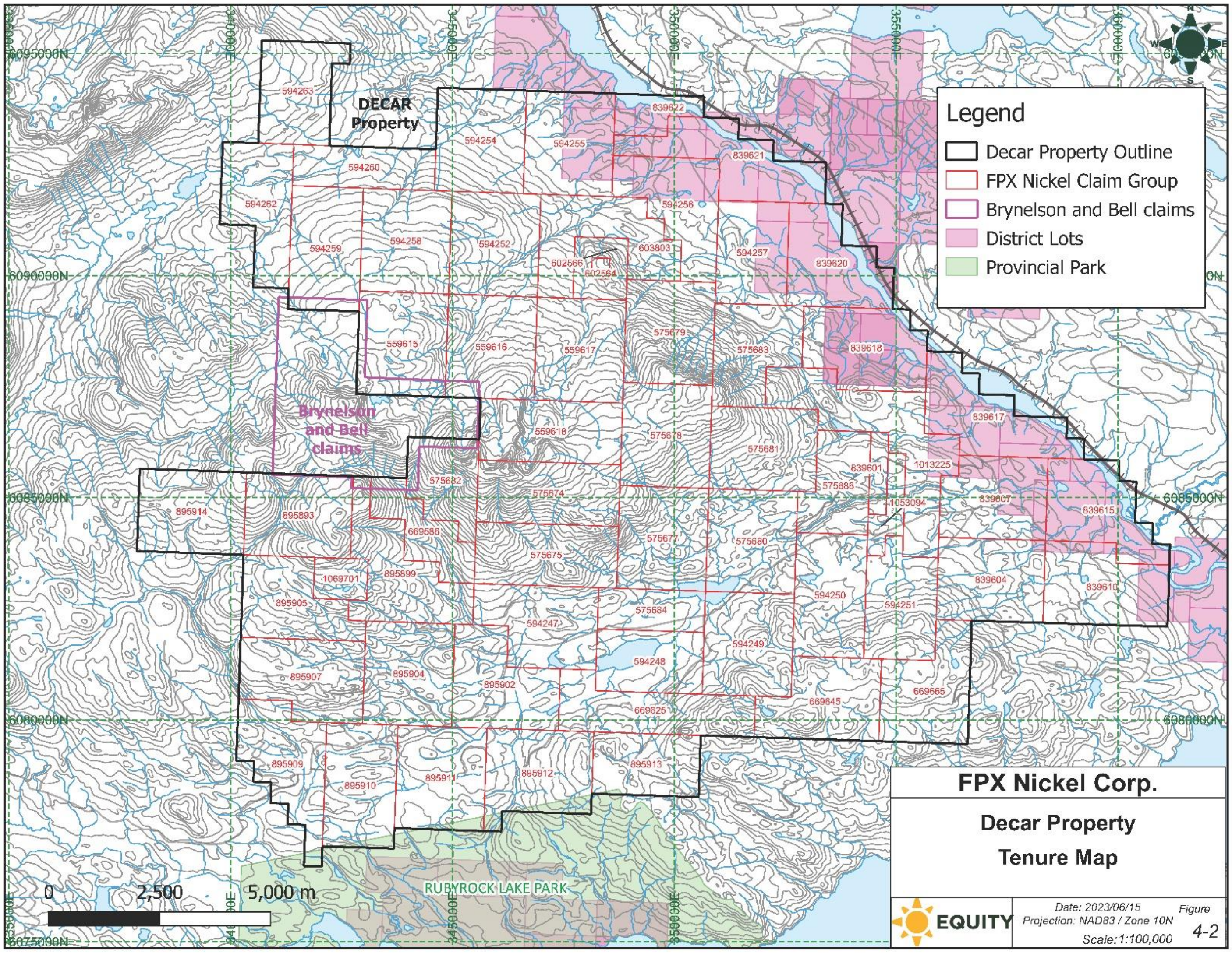
Source: Equity Exploration, 2023.

All claims are registered in the name of and are 100% owned by FPX. Cliffs Natural Resources Ltd (Cliffs) optioned the Decar Property in November 2009, and earned a 60% interest by incurring approximately US$22 M in exploration and related expenditures (FPX, 2014) but then sold its interest back to FPX in 2015 for a cash payment of US$4.75 M. This payment was made by a lender who earned a 1% NSR royalty over the Decar Property (FPX, 2015); this loan is fully repaid. The authors are otherwise unaware of any other debts, royalties, back-in rights or other agreements and encumbrances to which the property is subject.

No material environmental liabilities were noted by authors Voordouw and Flynn during a visit to the Decar Property in July 2022. The authors are otherwise unaware of any environmental liabilities or any other risks that may prevent FPX from carrying out future work.

Exploration programs that necessitate mechanical disturbance (e.g., drilling or trenching) and some ground-based geophysical methods require permits issued by the Ministry of Energy and Mines of the Government of BC.

Figure 4-2: Decar Property Tenure Map (A List of Claims is Provided in Appendix A)



Source: Equity Exploration, 2023.

To the author's knowledge, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

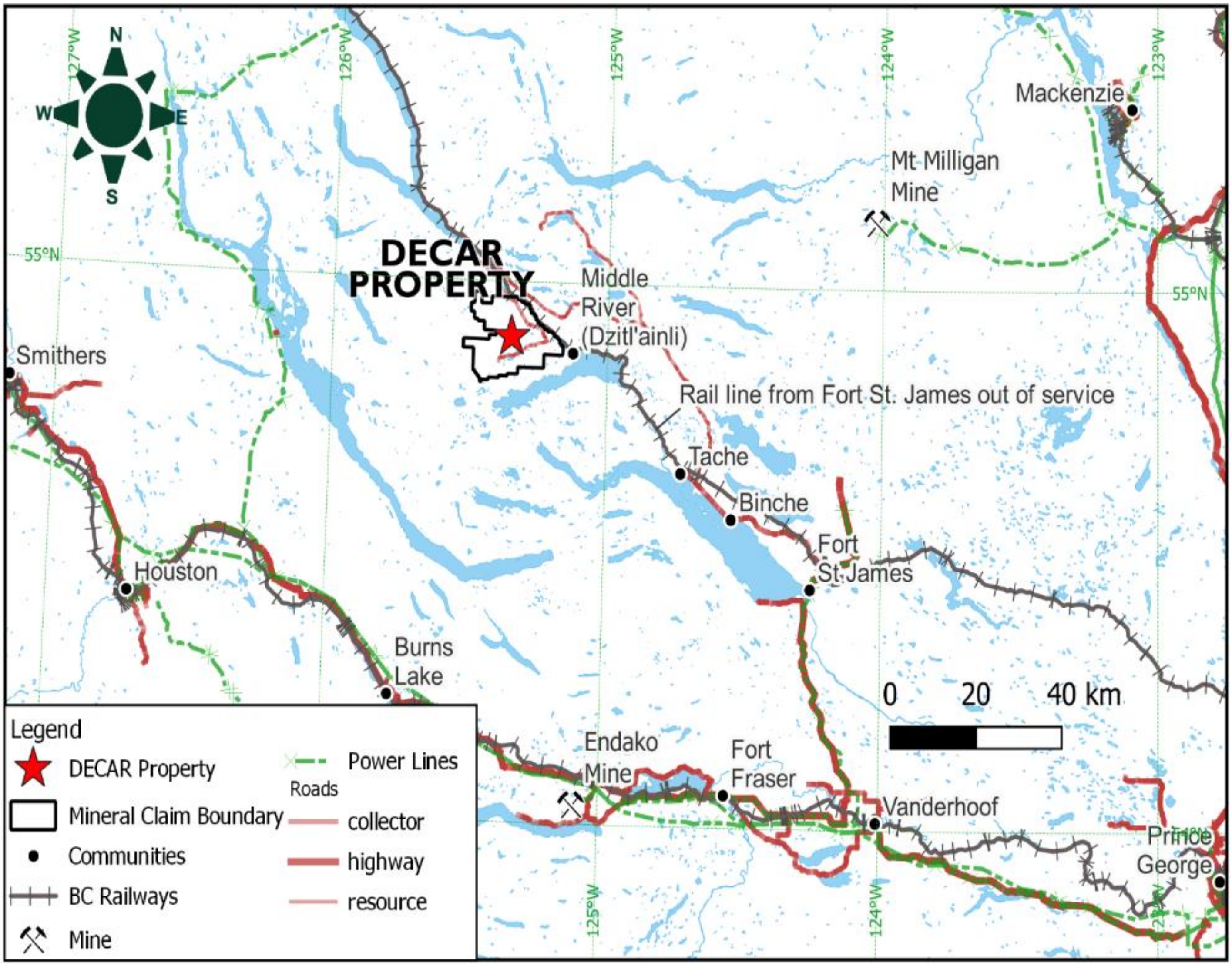
5.1 Physiography

The Decar Property lies in the sub-boreal spruce ecological zone within the rolling terrain of British Columbia's interior plateau (Figure 5-2). Elevation across most of the Property ranges from 900-1,400 m above sea level (masl), dropping to 700 masl near the Middle River and reaching a maximum of 1,983 masl at the peak of Mount Sidney Williams (also known as Tselkun, Wally's Mountain, and Red Rock). The property boundaries approach the Middle River in the northeast and Trembleur Lake in the south. The sub-boreal spruce ecological zone comprises dense coniferous forests that vary with elevation and localized micro-climates.

5.2 Accessibility

The Decar Property is road-accessible from the town of Fort St. James BC, over a 170 km long network of paved and gravel FSRs (Figure 5-1). The most direct route heads 2.3 km north of Fort St. James along Stuart Lake Highway/BC-27N to the Tachie Road, then follows the Tachie Road for 39 km to the Leo Creek FSR. The Leo Creek FSR is then followed northwest for 38.5 km, followed by 48 km on the Leo-Kazcheck FSR (or 300 Road), 2 km to Leo-Sakenichie FSR (or 900 Road) to cross the Middle River bridge and then 38 km on the Leo-Middle FSR (or 700 Road) to reach a 1 km long road leading to the Baptiste Deposit area.

Figure 5-1: Decar Property Access and Infrastructure Map



Source: Equity Exploration, 2023.

5.3 Climate

The Decar Property is subject to a northern temperate or sub-boreal spruce zone climate characterized by cold, snowy winters and warm summers. Environment Canada climate data for the Babine Lake/Pinkut Creek station, located approximately 50 km from the centre of the property, indicates daily average temperatures range from a low of -7°C in January to a high of 15°C in July (Environment Canada, 2022).

Precipitation ranges between 25 to 53 mm per month for the entire year, with the most precipitation occurring in June and the least in March and April. Snowfall can occur from October to April and average snow depths peak at 45 cm in

February. Ongoing environmental baseline studies include surface hydrology and atmospherics studies which will inform site-specific hydrometeorological assessments.

The physiography and climate at the Decar Property are amenable to year-round mineral exploration and development activities like diamond drilling. Geological mapping and geochemical sampling are best conducted from June to October when there is no snow cover.

5.4 Local Resources and Infrastructure

The closest municipality to the Property is Fort St. James, which has a population of 1,386 (Statistics Canada, 2021) and offers a range of services and supplies that include labour, gas stations, freight, heavy equipment rental, groceries, and hardware. The city of Prince George (population 89,490) lies 152 km by paved roads to the southeast of Fort St. James, and provides a broader range of services and supplies, along with daily commercial flights to Vancouver. The town of Smithers (population 5,316) lies 120 km due west of the property and can be accessed by FSRs and a seasonal barge that crosses Babine Lake. Smithers also offers a full range of services and supplies for mineral exploration, in addition to daily commercial flights to Vancouver.

Surface rights over the Decar Property are mostly owned by the Crown and administered by the Government of BC. Parts of several claims at the east end of the property overlap with District Lots (see Section 4) where exploration activity by FPX would require notification of the surface rights holder (if any). No exploration can be conducted on those parts of the Decar Property that overlap with Rubyrock Lake Provincial Park and, possibly, Mineral Reserve Site 326751.

There are several current- and past-producing mines in the area, including Mount Milligan, located 85 km east-northeast of the Decar Property, and Endako located 95 km to the south. Mount Milligan is a conventional truck and shovel, copper-gold, open pit mining operation with an on-site mill capable of processing 60 kt/d. The current life-of-mine (LOM) plan for Mount Milligan forecasts production through to 2033 (Borntraeger et al., 2022). Endako is an open pit molybdenum operation with a concentrator that has been on care and maintenance as of July 2015 (Centerra Gold, 2018).

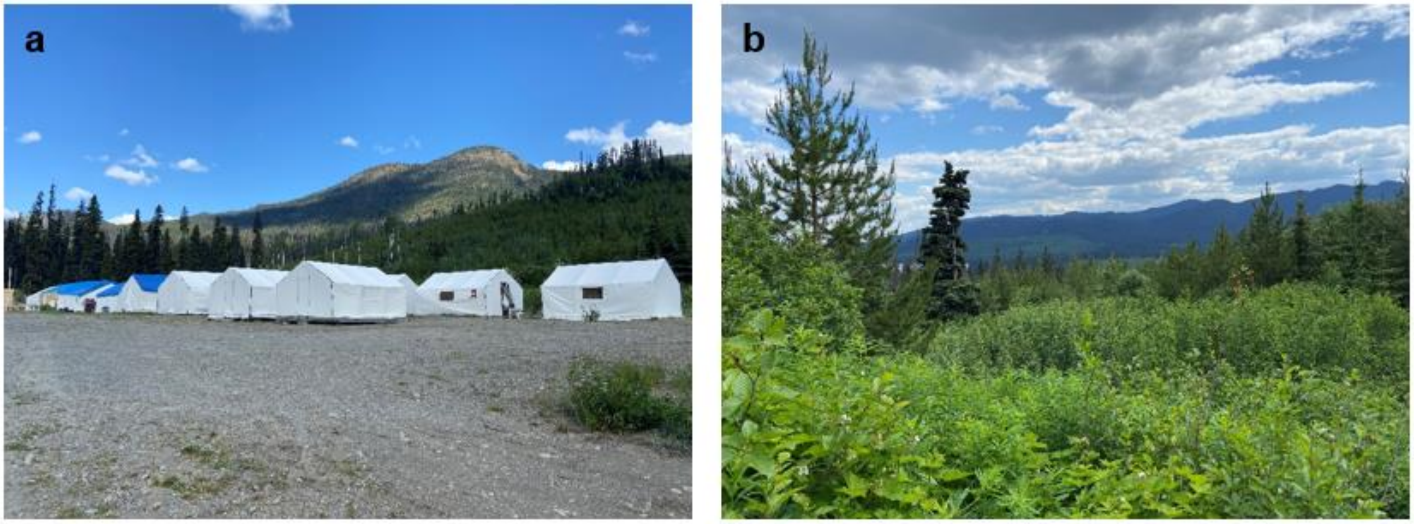
Obtaining the necessary environmental approvals and permits to operate a mine are described in Section 20 of this report.

FPX has historically operated a seasonal, road-accessible, industrial camp that consists of canvas tents placed on lumber tent platforms (Figure 5-2). Canvas tents are removed at the end of each fieldwork season.

CN Rail operates a railway through Smithers and Terrace to the port of Prince Rupert (Figure 5-1). An out-of-service rail line, also owned by CN Rail, follows the east bank of Middle River, and runs through the northeastern margin of the Property.

The nearest power corridor reaches 3 km to the Decar Property and serves the Middle River Village. BC Hydro’s Glenannan Substation connects to a 500 kV trunk line and is located 90 km south-southeast of the property.

Figure 5-2: Terrain at the Baptiste Deposit



Note: Photos looking (a) north from the southeastern end of the Baptiste Deposit, across the 2022 temporary Decar camp and towards more rugged elevated terrane in the background, and (b) from the central part of the Baptiste Deposit to rolling and forested terrain in the southeast part of the Property.
Source: Equity Exploration, 2022.

6 HISTORY

6.1 Early Chromium, PGE, and Gold Exploration

Exploration activity on or immediately adjacent to the Decar Property has mostly focused on ultramafic-hosted mineral deposit types, such as chromite and PGE from the mid-1970s to mid-1980s, listwanite-hosted gold from approximately 1987 to 1994, and the strongly magnetic and high-density mineral awaruite from the mid-1990s to today.

Chromite pods are a source of chromium and, potentially, platinum group elements (PGE) and were first described in the Decar area by the Geological Survey of Canada (Armtrong, 1949). This work discovered nine chromite occurrences within the Trembleur ultramafite unit, the largest of which is exposed over 1.5 km by 7.5 km and contains >50% chromite (Armtrong, 1949). In 1975, prospecting near what is now the centre of the Decar Property found mostly disseminated chromite grading 0.2-0.4% Cr, with one select sample of chromite-rich dunite returning 9.8% Cr (Stelling, 1975). Subsequent work in the same area by Mountaineer Mines re-discovered the 1.5 km x 7.5 km zone identified by the Geological Survey of Canada, returning assays of 17.8% and 38.9% Cr on select rock samples but otherwise failing to locate larger chromite bodies (Guinet, 1980).

Three years later, Northgane Minerals Ltd subcontracted Western Geophysical Aero Data Ltd to fly a 310 line-km very low frequency electromagnetic (VLF-EM) and magnetom survey with the aim of defining the extent of ultramafic rocks and identifying trends that could be favourable for chromite mineralization (Pezzot and Vincent, 1982). This survey identified highly magnetic areas of possibly serpentinized peridotite and dunite, showing that at least three chromite showings are associated with very high magnetic response (Pezzot and Vincent, 1982); however, no further chromite-focused exploration was completed.

The discovery of economic gold deposits hosted in “listwanite” – a partially silicified, carbonate-altered, ultramafic rock – elsewhere in BC brought renewed exploration interest to the Decar area. Consequently, M. Ursula Mowat established a property that overlaps with what is now the central and western half of the Decar Nickel District, and, in 1987, began exploration for gold-hosted listwanite through an option agreement with Lacana Mining Corporation (Mowat, 1988a). The next year, surface sampling and trenching of seven listwanite zones returned gold (Au) enriched channel samples (Mowat, 1988b). Coincident prospecting, in what is now the southern part of the Decar Property, returned ultramafic rock samples with 80-90 ppb Au (Forbes, 1988).

Diamond drilling of listwanite prospects in 1990-1994 (1,541 m in 22 drillholes) were financed through option agreements between Ursula Mowat and Viceroy Resource Corporation (Mowat, 1990; Mowat, 1991; Mowat, 1994), Minnova Incorporated (Mowat, 1991), and Teryl Resources Corporation (Mowat, 1994). Six of the first seven holes drilled into listwanite prospects returned intercepts of 1-6 g/t Au over intervals ranging from 0.4 to 9 m (Mowat, 1990). This would turn out to be the most successful program. The next year, another five holes were drilled on the Stibnite and Upper listwanite zones but returned mostly disappointing results (Mowat, 1991). In 1994, another 10 holes tested EM conductors, soil anomalies, listwanite zones, and/or IP features (Mowat, 1994) with drilling generally failing to intersect gold mineralization or explain geophysical anomalies.

Following a brief lull, exploration work resumed in 1998 to focus on the previously unexplored West Peak area as well as strongly talc-altered ultramafic rocks near Baptiste Creek (Mowat, 1998). Results were used to suggest that glacial detritus could be masking a porphyry and/or gold system on the property. The following year, a three-phase work program was used to define, and then follow-up on, weak Au and PGE anomalies on the Mid claim (Mowat, 1999). Mappers identified additional listwanite and talc-rich alteration zones with elevated arsenic and nickel, as well as glassy volcanic rocks with weakly anomalous platinum group elements (PGE). The following year, new outcrops of ultramafic were discovered in the West Peak area, although sampling returned no elevated base or precious metals (Mowat, 2000). From 2002-2004, a small surface program outlined new listwanite occurrences with coincident Au-arsenic soil anomalies (Mowat, 2002), new outcrops of listwanite and serpentinized peridotite in clear cuts (Mowat, 2004), and Au-enrichment in soil (Mowat, 2005). A small-scale soil and rock sampling program completed in 2006 returned negligible results (Mowat, 2007).

In 2007, AMARC conducted a significant silt sampling campaign on claims that lie mostly west of the current Decar Property, but also overlap with the southwestern-most of these claims. This work returned anomalous values for molybdenum, copper, and zinc (Ditson et al., 2008).

6.2 Awaruite Exploration

Awaruite in the Decar area was first recognized by Whittaker (1983) while completing a PhD thesis on several belts of ultramafic rocks in central BC. In 1996, geochemical and petrographic work on ultramafic samples from what is now the centre of the Decar Property showed that nickel is hosted in awaruite and other low-sulphur nickel minerals, including heazlewoodite, bravoite and pentlandite (Mowat, 1997b; Mowat, 1997a). It was recognized that since awaruite, magnetite and chromite were all magnetic, a concentrate could potentially be produced by magnetic separation. This work was followed up with a sampling and metallurgical program that showed awaruite may be economically extractable through a simple grind and magnetic separation process (Mowat, 1997c). The 1999 program included metallurgical test work on different grind sizes, with results showing that a 150 mesh grind produces higher Ni values than 100 mesh (Mowat, 1999).

Besides rudimentary prospecting, petrographic study, and metallurgical work, little further work was completed on awaruite mineralization until 2007 when First Point Minerals Corp., a precursor to FPX, staked 15 claim that now form the core of the Decar Property (Voormeij and Bradshaw, 2008). The Decar Property then grew to 33 claim by 2009 (Britten, 2010), 37 claim by 2010 (Britten and Rabb, 2011), 59 claim by 2012 (Ronacher et al., 2012a) and then to its current 62-claim size by 2018. Work by FPX and its precursor company, First Point Minerals Corp., is further described in Sections 9 and 10.

FPX optioned the Decar Property to Cliffs in November 2009, who earned a 60% interest in the Property by incurring approximately US$22 M in exploration and related expenditures (FPX, 2014), including completion of a PEA (McLaughlin et al., 2013). In 2015, FPX re-acquired Cliffs’ 60% ownership interest with a cash payment of US$4.75 M, provided by a lender who earned a 1% NSR royalty over the Decar Project (FPX, 2015); this loan is now fully repaid.

6.3 Historical Resource Estimates

Historical mineral resource estimates were completed by FPX in 2012, 2013, 2017, and 2020 (Table 6-1). All four of these estimates were completed in accordance with NI 43-101 reporting standard and were calculated for a cut-off grade of 0.06% DTR nickel. Nonetheless, these historical resource estimates have not been verified by the authors, are not considered relevant, and should not be relied upon for any use. A qualified person (QP) has not done sufficient work to classify these historical estimates as current mineral resources and FPX is not treating any of the historical estimates as current mineral resources. An updated resource for The project is provided in Section 14 of this report.

No other significant historical mineral resource estimates have been reported for the property.

Table 6-1: Historical Resource Estimates for the Baptiste Deposit

Year	Resource Category	Cut-Off DTR Ni (%)	Quantity (tonnes)	Grade DTR Ni (%)	Contained Metal DTR Ni (tonnes)	Source
2012[a]	Inferred	0.06	1,197,000,000	0.113	1,352,610	Ronacher et al., 2012b
2013[b]	Indicated	0.06	1,159,510,000	0.124	1,437,792	Ronacher et al., 2013; McLaughlin et al., 2013
	Inferred	0.06	870,400,000	0.125	1,088,000	
2018[c]	Indicated	0.06	1,842,645,000	0.123	2,271,200	Voordouw and Simpson, 2018
	Inferred	0.06	390,788,000	0.115	448,200	
2020[d]	Indicated	0.06	1,995,873,000	0.122	2,434,965	Grandillo et al., 2020
	Inferred	0.06	592,890,000	0.114	675,895	

Source: Equity Exploration, 2022.

[a, b]: QP for estimate is Mr. Jason Baker, P.Eng. of Caracle Creek International Consulting Inc. Tonnes were rounded to nearest 10,000 and grade was rounded to three significant digits. The estimate was made using ordinary kriging.

[b]: Mining cost = US$3.43/t, mining recovery = 97%, mine dilution = 8%, processing cost = US$4.02/t, process recovery = 82%, Ni price = US$24,244/t. Mineral resources were constrained by a whittle pit.

[c, d]QP for estimate is Mr. Ronald Simpson of GeoSim Services. Indicated resources drilled on 200 x 200 m spacing, inferred is 300 x 300 m. Estimate was made using ordinary kriging.

[c]: Mineral resources were constrained by an optimized pit shell using the following assumptions: Mining cost = US$2.72/t, mining recovery = 97% DTR Ni, processing cost = US$5.32/t, process recovery = 82% DTR Ni, Ni price = US$6.00/pound.

[d]: Mineral resources were constrained by an optimized pit shell using the following assumptions: Mining cost = US$2.75/t, mining recovery = 97% DTR Ni, processing cost = US$4.00/t, process recovery = 85% DTR Ni, Ni price = US$6.35/pound.

6.4 Production

No ore production has been reported for the property.

7 GEOLOGICAL SETTING AND MINERALIZATION

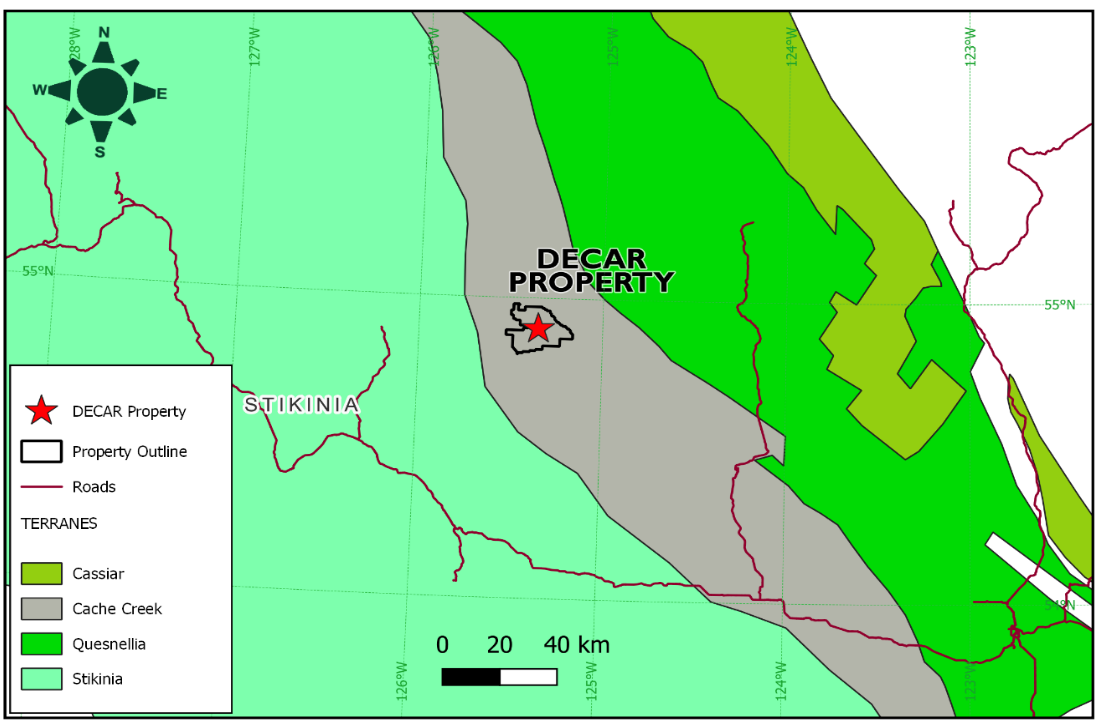
7.1 Regional Geology

Mineralization on the Decar Property is hosted in serpentinized ultramafic rocks of the Cache Creek terrane. The following sections establish the regional– to property-scale context of these ultramafic rocks and associated awaruite mineralization.

The Decar Property lies entirely within the Cache Creek terrane of British Columbia (Figure 7-1), which consists of Carboniferous to Jurassic rocks that formed in an oceanic and/or back-arc setting (Monger, et al., 1991). This terrane stretches from the southern Yukon to its type locality in southern BC, and consists of radiolarian chert and argillite, shallow-water carbonate, ophiolite (basalt, gabbro, ultramafic rocks) and calc-alkaline volcanic to volcano-sedimentary rocks. Serpentinized ultramafic rocks of ophiolitic-affinity are the host of awaruite mineralization on the property.

The Cache Creek terrane is bound by regional-scale faults that separate it from the Stikine and Quesnel terranes to the west and east respectively and shows significant internal stratigraphic and structural disruption (Monger, et al., 1991). The western margin of the Cache Creek terrane occurs approximately 10 km west of the property and is formed by the Thudaka-Finlay-Ingenika-Takla fault system. The eastern margin occurs approximately 10-20 km east of the property and is formed by the Thibert-Kutcho-Pinchi fault system.

Figure 7-1: Regional Geological Setting of the Decar Property Showing the Cache Creek (CC), Stikine (ST) and Quesnel (QN) Terranes.



Source: Equity Exploration, 2022

Both the Thudaka-Finlay-Ingenika-Takla and Thibert-Kutcho-Pinchi fault systems are regional-scale structures that are mostly subvertical and northwest to north-northwest striking. An estimated 11 km of dextral movement had occurred on the Thibert-Kutcho-Pinchi fault system by the Upper Cretaceous and was followed by another 170 km of movement on the north-south trending Thudaka-Finlay-Ingenika-Takla fault system into the Eocene (Gabrielse, 1985). This protracted history of accretion and lateral transport resulted in significant faulting internal to the Cache Creek terrane and likely channeled the fluids that caused serpentinization of ultramafic rocks and awaruite mineralization (Britten, 2016).

The regional-scale faults that bound the Cache Creek terrane are offset by northeast trending dextral strike-slip faults like the Trembleur Lake and Tildesly Creek faults (Britten and Rabb, 2011).

7.2 Regional Metallogeny

Mineral deposits in the Cache Creek terrane include several types within the ultramafic ophiolitic rocks, as well as Noranda- and Kuroko-type copper-lead-zinc volcanogenic massive sulphide (VMS), molybdenum-copper porphyry, vein-hosted gold and surficial placer gold. Deposit types formed in ultramafic rocks include the awaruite prospects described in this report (see also Britten, 2016), Alaskan-type Ni-copper-PGE, podiform chromium, jade/nephrite, listwanite-hosted gold, talc-magnesite, cryptocrystalline magnesite veins, magmatic Ni-Cu sulphide, and asbestos.

The bulk of the historical and current mineral production from Cache Creek rocks is from relatively small-scale jade/nephrite, placer gold, and industrial mineral operations. The adjacent Stikine and Quesnel terranes host significant Cu-Au porphyry deposits like the Mount Milligan and Endako mines, which lie 85 km northeast and 95 km south of the Decar Property, respectively.

7.3 Property Geology

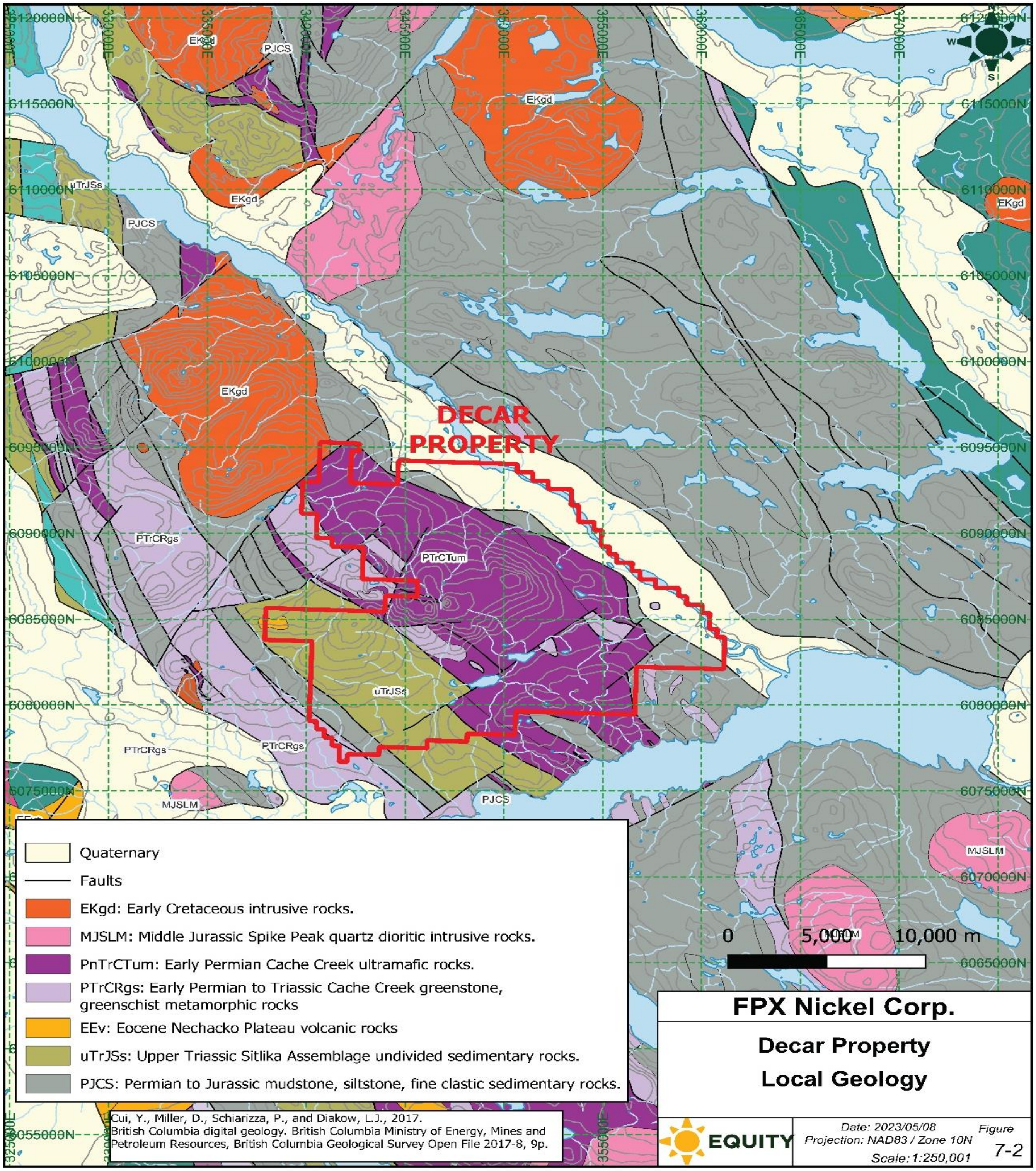
The Cache Creek terrane that underlies the Decar Property is subdivided into (from southwest to northeast) the Rubyrock igneous complex, Sitlika assemblage, Trembleur ultramafite, and Sowchea succession (Cui, et al., 2017; Figure 7-2, Figure 7-3). These subdivisions are structurally intercalated by faults internal to the Cache Creek terrane.

Awaruite mineralization occurs entirely within the Trembleur ultramafite and is correlated to increased serpentinization.

7.3.1 Lithology

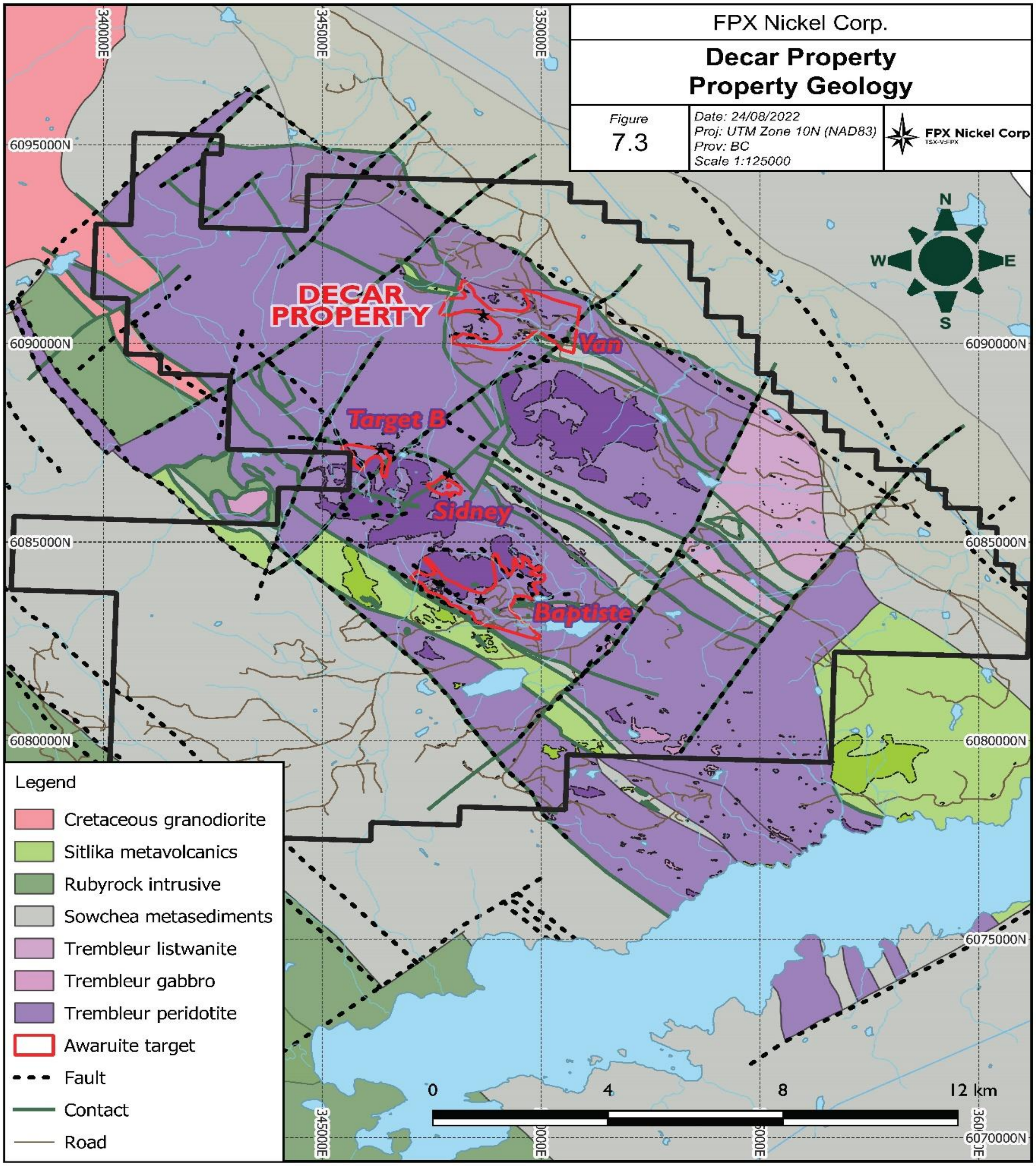
The Rubyrock igneous complex consists of upper Paleozoic to Triassic gabbro, basalt, diabase and micrograbbro (Struik, et al., 2001) that could be analogous to the upper part of a typical ophiolite succession (see Boudier and Nicolas, 1985). These rocks occur mostly immediately southwest of the property with intercalations occurring along its contact with the Trembleur ultramafite.

Figure 7-2: Local Bedrock Geology of the Decar Property.



Source: Equity Exploration, 2022

Figure 7-3: Property-Scale Bedrock Geology, Structure, and Mineral Deposits of the Decar Property.



Source: Equity Exploration, 2022

The Sitlika assemblage underlies the southwestern portion of the property and consists of carbonaceous argillite, black phyllite, mudstone, slate, limestone, chert, and chloritic phyllite that are broadly comparable to the regional-scale assemblage as described by Struik, et al (2001). This unit is fault bound along much of its strike length, tectonically interleaved with Trembleur ultramafite, and has bedding that is mostly steeply dipping to subvertical.

The Trembleur ultramafite forms the core of the Decar Property with additional occurrences to the south and north. Rock types consist mostly of peridotite with lesser amounts of dunite, pyroxenite and gabbro (Struik, et al., 2001; Britten, 2016) that are typically 60-100% serpentinized. Pre-alteration mineralogy is estimated at 65-80% olivine, 20-30% orthopyroxene, <5% clinopyroxene and <0.5% chromite (Britten, 2016). The Trembleur ultramafite is interpreted as the mantle and lower crustal portion of a typical ophiolite succession and hosts awaruite mineralization.

Core logging describes peridotite as typically 80-100% serpentinized with less abundant intervals of “massive peridotite” showing 60-80% serpentinization. Both types of peridotite average 40 wt% MgO, 8 wt% Fe_2O_3, and 2,500 ppm Ni. The abundance of secondary serpentine is manifested in rock colour, with the least altered peridotite appearing medium grey, whereas those with greater than 95% serpentine are dark green to black-brown. Dunite occurs throughout the ultramafite, typically comprising greater than 95% serpentinized olivine (Britten, 2016) and averaging slightly higher contents of whole rock MgO (42 wt%) and Ni (2,600 ppm) relative to peridotite. Both peridotite and dunite host awaruite although grades within the dunite are generally lower.

Cataclastic peridotite is marked by higher fracture density and, on average, hosts the largest awaruite grains. Average MgO (39 wt%) and Ni (2,000 ppm) are lower than less deformed ultramafic rocks. Mylonitized peridotite also hosts relatively large awaruite grains and lower whole rock MgO (36 wt%) and Ni (2,000 ppm) contents, as well as weakly elevated Ca, Ti, Sr, V, Y, and Zr. Hornfelsed peridotite occurs next to gabbro and altered dikes, forming dark, very fine grained selvages that are typically approximately 1-5 m in core width. These selvages are strongly fractured and contain relatively scarcer and smaller awaruite grains, as well as relatively low MgO (37 wt%) and Ni (2,000 ppm) contents along with weakly elevated Al, Si, K, Na, Sc, Sr, V, Y, and Zr.

Intervals logged as fault zone show similarities to cataclastic peridotite, including high fracture density, relatively large awaruite grains and high Davis Rube Recoverable (DTR) Ni contents, and similar average MgO (38 wt%) and Ni (2,100 ppm) contents.

Carbonate-altered ultramafic rocks have similar MgO (37 wt%) and Ni (2,100 ppm) contents to deformed ultramafic rocks but significantly lower DTR Ni. Whole rock CaO contents are only weakly elevated, suggesting that the predominant carbonate species is likely siderite and/or magnesite. Listwanite has higher CaO and loss on ignition (LOI).

The Trembleur and Sitlika units are cut by gabbro and altered dikes that show a close spatial association with each other as well as fault and carbonate alteration zones. Gabbro forms massive, fine- to medium-grained stocks and northeast to east striking dikes. Altered dikes include rodingite (garnet-pyroxene-rich rocks) and/or other rock types with serpentine, carbonate, chlorite, hematite, silica, albite and/or fuchsite. Whole rock geochemistry suggests that altered dikes are derived from gabbro dikes with minimal addition or removal of elements. Most dikes are dilutive with respect to DTR Ni grades.

The Sowchea succession forms the northeastern wall rock to the Trembleur unit and is tectonically interleaved with it along contact zones. Rock types consist of fine-grained clastic sedimentary rocks (e.g., mudstone, fine sandstone) and their metamorphic derivatives, as well as chlorite schist and metabasalt (Logan, et al., 2010). This unit is interpreted as equivalent to the pillow basalt and deep-sea sedimentary rocks that comprise the top of the typical ophiolite column.

Sowchea rocks were referred to as North Arm succession (after Schiarizza and MacIntyre, 1999) in some of the previous technical reports (e.g., McLaughlin, et al., 2013; Ronacher, et al., 2013).

A 12 km by 8 km, 104 Ma, granodiorite to quartz diorite intrusion extends into the northwestern-most part of the property surficial mapping by the GSC (Plouffe, 2000) indicates that most of the property, especially at intermediate elevations, is covered by Quaternary till blanket and veneer. Higher elevations consist of bedrock outcrops covered by slope colluvium and talus whereas lower elevations are covered by glaciolacustrine deposits and recent lacustrine and alluvial sediments. Diamond drilling indicates that true overburden depths range from 0 to 50 m across the drilled areas, with a median depth of 7 m.

7.3.2 Structural Geology

At least two generations of structures occur on the Decar Property, here broadly subdivided into older northwest-striking structures and younger northeast ones (Figure 7-3).

Northwest (NW) striking structures are generally subvertical, ≤100 m wide, one to tens of kilometers long, and overprint all Cache Creek units on the Decar Property. These structures are particularly well developed at major unit contacts but also form discrete belts that are internal to these units. Most of them are defined by a core of cataclasite, mylonite, and/or fault breccia that passes symmetrically outwards into decreasing intensities of penetrative deformation fabric. Such fabrics suggest ductile deformation although localized occurrences of slickensides and/or gouge suggest that some NW-trending structures underwent later, brittle, reactivation. NW-striking structures were likely involved in the two most significant tectonic events that affected the Cache Creek terrane, namely the Middle Jurassic obduction of Cache Creek onto Quesnellia and the long-lived transform faulting in the Cretaceous to Eocene.

NW-striking structures within the Trembleur ultramafite also appear to control serpentinization and related awaruite mineralization. The Baptiste Deposit, Sid target, and Van target, for example, are all elongated along a NW-SE direction and bound by subvertical, NW-SE striking, 50-100 m wide, belts of strong penetrative deformation fabric (Britten, 2016). This relationship suggests that NW-striking structures channeled the fluids that caused serpentinization of the Trembleur ultramafite and resultant awaruite mineralization. Petrographic study shows that NW-striking structures formed through multiple breakage events, resulting in pseudo-brecciated peridotite cross-cut by hairline fractures and serpentine veins that formed both before and during serpentinization (Britten and Rabb, 2011; Britten, 2016).

There are several younger NE-trending structures on the property, the most prominent of which comprises a subvertical fault that appears to have caused up to 1,500 m of dextral offset of the Trembleur unit (MacIntyre and Schiarizza, 1999) in the southern part of the Decar Property. The surface expression of the Baptiste Deposit also suggests a possible secondary NE-directed control.

7.3.3 Alteration

Serpentinization and Mg-Fe carbonate alteration are the predominant alteration types within the Trembleur ultramafite. Serpentinization is the most widespread, with all ultramafic rocks within the Trembleur unit estimated to be 60-100% serpentinized. On the Decar Property, serpentinization is defined by the replacement of olivine and orthopyroxene with antigorite and lizardite rather than chrysotile (Britten, 2016). Magnetite and awaruite form as part of the serpentinization process.

More recent work (Steinthorsdottir, 2021) identified four generations of serpentinite alteration referred to as Serp 1A, 1B, 2, and 3. Stage 1 is defined by relatively low-temperature assemblages comprising (Serp 1A) earlier stage massive lizardite + brucite veins cut by (Serp 1B) later-stage ribbon and mesh-textured lizardite. Stage 2 consists of higher temperature, interlocking and vein textured, antigorite. Stage 3 is relatively rare, preceded by carbonate alteration, and defined by cross-fibre chrysotile ± antigorite veins. Awaruite growth occurred during Stages 1 and 2.

Mg-Fe carbonate alteration forms a continuum from carbonate-only to carbonate-silica (i.e., "listwanite") assemblages, with both types antithetic to awaruite mineralization. Mg-Fe carbonate altered rocks have average CaO contents (1.2 wt%) that are only slightly higher than peridotite (1.0 wt%), suggesting that most carbonate is likely siderite or magnesite. Increasing alteration intensity is also marked by decreasing magnetism and progression from selective replacement texture to pervasive.

Listwanite alteration is typically more pervasive than carbonate-only alteration as well as more texturally and magnetite destructive. Whole rock geochemistry indicates relatively high average contents of CaO (1.8 wt%) and LOI (18.9 wt%), which is consistent with stronger alteration. Silica precipitates from the near-total carbonate alteration of the parent ultramafic rocks. On the property, listwanite is spatially associated with fault and/or unit contact zones, and possibly feldspar porphyry intrusions. Listwanite hosts pyrite, rare chalcopyrite, and trace amounts of gold (Britten and Rabb, 2011) and was explored as part of early gold-focused programs (see Section 6.1).

7.4 Property Mineralization

Historical exploration on the property has focused on three deposit types: (a) awaruite in serpentinized ultramafic rock, (b) listwanite-hosted gold, and (c) chromite pods in ultramafic. Each of these is briefly described below even though, at this time, only the awaruite deposits are potentially economic.

Awaruite deposits are formed by the serpentinization of magmatic olivine that leads to the liberation of nickel and iron (Britten, 2016) and subsequent formation of awaruite. Awaruite occurs as disseminated grains throughout the entire extent of the peridotite on the Decar Property, but four zones of more abundant mineralization and larger grain size have been identified and named; Baptiste, Sid, B, and Van. The Baptiste Deposit and Van target are the most advanced of these zones, with both tested by numerous drill holes (see Section 10). The Sid and B targets are defined through surface mapping, sampling and/or diamond drilling, with three holes drilled at Sid target and one drilled at B target. High-grade awaruite zones trend NW-SE, parallel to lithological contacts and older fault structures.

The Baptiste Deposit is currently defined for approximately 3,000 m in a west of northwest to east of southeast direction, and for 400-1,500 m from north-northeast to south-southwest. Diamond drilling shows that mineralization typically extends to vertical depths of at least 200-300 m, with 13 of 28 holes drilled deeper than 300 m and ending in mineralization. The Deposit is in fault contact with Sitlika metavolcanic rocks to the southwest and grades into massive peridotite with lower-grade mineralization to the north and northwest.

The Van target lies 7.4 km north of the Baptiste Deposit and has a surface expression of approximately 2,200 x 700 m, with the eastern part of this target defined by 2021 and 2022 drilling (1,000 x 700 m) and the western half by rock sampling (1,200 x 700 m). The Van target is fault bound by Sowchea metasediments to the northeast and along part of its southwest boundary, with the remaining southwestern boundary formed by massive peridotite.

The Sid and B targets occur 3.0 km north-northwest and 4.5 km northwest of the Baptiste Deposit, respectively, and were also initially defined by rock sampling. Drill holes in both targets are sufficient only to demonstrate that Baptiste-like mineralization occurs there as well, but insufficient to determine the length, width, depth, and continuity of the mineralization.

Exploration for listwanite-hosted gold deposits peaked in 1990-1994, with 1,541 m of diamond drilling over 22 holes to test the most prospective zones (Mowat, 1990, 1991, 1994). By 1994, drilling had identified at least 17 listwanite zones on or around the current Decar Property. Pathfinder elements for listwanite-hosted gold occurrences include iron, arsenic, lead, copper, zinc, nickel, cobalt and antimony. Possible origins of the gold-listwanite association include (1) carbon dioxide and hydrogen sulphide immiscibility with attendant gold deposition, (2) reduction of the mineralizing fluid by gold deposition, (3) sulphide precipitation promoted by Fe-rich lithologies, and (4) precipitation of silica, pyrite, arsenide and gold when an acidic gold-bearing solution enters reduced and alkaline carbonatized rocks (Kerrich, 1989; Dussel, 1985).

Mapping by the GSC identified nine chromite pods in the Trembleur ultramafic rocks within or near the Decar Property, with follow-up work suggesting these pods are generally too small to be economic. These occurrences are examples of podiform chromite that formed in the ultramafic part of an ophiolite complex.

8 DEPOSIT TYPES

8.1 Deposit Model

Disseminated awaruite (Ni_3Fe) forms an unusual nickel deposit type, with occurrences on the Decar Property comprising the most advanced projects of this type in the world (Britten, 2016). Terrestrial awaruite was first described in heavy black sand from the South Island of New Zealand (Frost, 1985; Ulrich, 1980) and has since been found as a minor component in altered ultramafic rocks all over the world. It forms during serpentinization of peridotite, whereby nickeliferous olivine is altered to serpentine minerals and awaruite + magnetite under conditions of low oxygen fugacity (Frost, 1985). A general unbalanced reaction that illustrates this mineralogical and metal exchange is as follows (from Britten, 2016):

$$4(Mg, Fe, Ni)_2SiO_4 + 2H_2O \leftrightharpoons 2Mg_3Si_2O_5(OH)_4 + Fe_3O_4 + Ni_3Fe + H_2(g)$$

The alteration of olivine-rich ultramafic rocks to 60-80% serpentine includes a gain of 10-14 weight% H_2O that results in a density decrease from approximately 3.3-3.4 g/cm^3 to 2.7 g/cm^3 and a volume increase of 18% to 55% (Britten, 2016).

A recent overview of the awaruite deposits hosted in the Cache Creek terrane (Britten, 2016) suggests that a key part of the mineralization forming process was a prolonged period of post-accretionary transpression, which resulted in significant strike-slip displacement and, more importantly, ingress of relatively clean and possibly oxygenated meteoric water. Deformation generated high porosity zones up to several hundreds of metres in width that are now marked by foliation, crackle breccia and microfracture textures. Subsequent processes then necessary to produce awaruite included the hydration of olivine to serpentine minerals, ingress of water with low sulphur and CO_2 activity, oxidation of iron to produce magnetite, the maintenance of low oxygen fugacity and, eventually, addition of H_2 through reduction of Fe and Ni. Hydration at temperatures of <100 to 200°C is likely capable of producing fine-grained awaruite (<20 µm) in association with low-temperature serpentine minerals (e.g., lizardite, chrysotile) and brucite (Britten, 2016). Higher temperatures (200 to >400°C) are probably necessary to form the larger grains like those on the Decar Property that are associated with antigorite. The highest temperature (>450°C) conditions produce the highest amount of magnetically-recovered awaruite, in association with the metamorphism of serpentine and magnetite to olivine and diopside (Britten, 2016).

Besides the Decar Property, other awaruite occurrences occur in the northern outcropping areas of the Cache Creek terrane (see Figure 7-1) and in the Dumont Deposit of Québec, Canada. Prospects in the northern Cache Creek terrane include Orca, Wale, Letain, and Mich, and are similar to those at Decar (see Britten, 2016). At the Dumont Deposit, awaruite occurs as pervasively disseminated grains, between <50 to 400 µm in size, hosted in serpentinite and spatially associated with magnetite and chromite (Staples, et al., 2013). Although sulphides are widespread in the Dumont deposit, there are zones where predominantly awaruite is present. Minor abundances of awaruite also occur together with nickel and copper sulphide in the Duluth complex of Minnesota, USA, and appears to be of magmatic, rather than secondary, origin.

Awaruite is highly magnetic and dense (ρ = 8.6 g/cm^3) and is therefore amenable to concentration by magnetic and gravity separation. Awaruite has active surface properties, and will respond to froth flotation, much like sulphide minerals, following conditioning to remove any passivated surface layer. In addition, the ultramafic tailings from awaruite concentrate production could potentially be used for CO_2 sequestration (e.g., Vanderzee et al., 2018).

Because metallurgical properties play such a vital role in the economics of awaruite projects, the grades are presented as Davis Tube Recoverable (DTR) nickel. The Davis Tube consists of an inclined water-filled tube placed between electromagnets (Svoboda, 2004) and is used to split finely-ground powder into magnetic and non-magnetic fractions, with the magnetic fraction subsequently assayed separately by XRF.

DTR nickel is calculated as follows:

$$\mathrm{DTR\ Ni} = \left(\mathrm{wt\%\ NiO}_{Mags} \times 0.7858\right) \times \frac{(\mathrm{Weight\ magnetic\ fraction})}{(\mathrm{Weight\ magnetic\ fraction} + \mathrm{nonmagnetic\ fraction})}$$

Data required to calculate DTR nickel is provided by the analytical lab, which besides reporting weight percent nickel oxide in the magnetic fraction (wt% NiO) also reports the weights of the magnetic and non-magnetic fractions split with the Davis Tube.

9 EXPLORATION

FPX staked approximately half of what is currently the Decar Property in 2006 and then conducted four years of additional staking, geological mapping and sampling, geophysical surveys, and metallurgical testwork before optioning The project to Cliffs in 2009. Cliffs then funded exploration on the Property from 2010 to 2013, publishing a maiden resource for the Baptiste Deposit in 2012 (Ronacher et al., 2012) followed by an updated resource (Ronacher et al., 2013) and PEA in 2013 (McLaughlin et al., 2013). Since 2013, FPX has completed additional mapping and surface sampling work, in addition to diamond drilling in 2017 and 2021.

This section describes geological mapping, surface sampling and geophysical surveys that have been carried out by FPX since 2006. Descriptions include procedures and parameters used to execute the work, sampling methodology and QA/QC, as well as a summary of significant results and interpretation. Diamond drilling is summarized in Section 10.

9.1 Surface Geological and Geochemical Surveys

Geological mapping, rock sampling and petrographic work done on the Decar Property is summarized in Table 9-1, with the sample totals shown derived from the FPX database. Mapping and rock sampling effectively define the Baptiste, Sid, B, and Van targets on the Property. Sampling locations are shown in Figure 9-1.

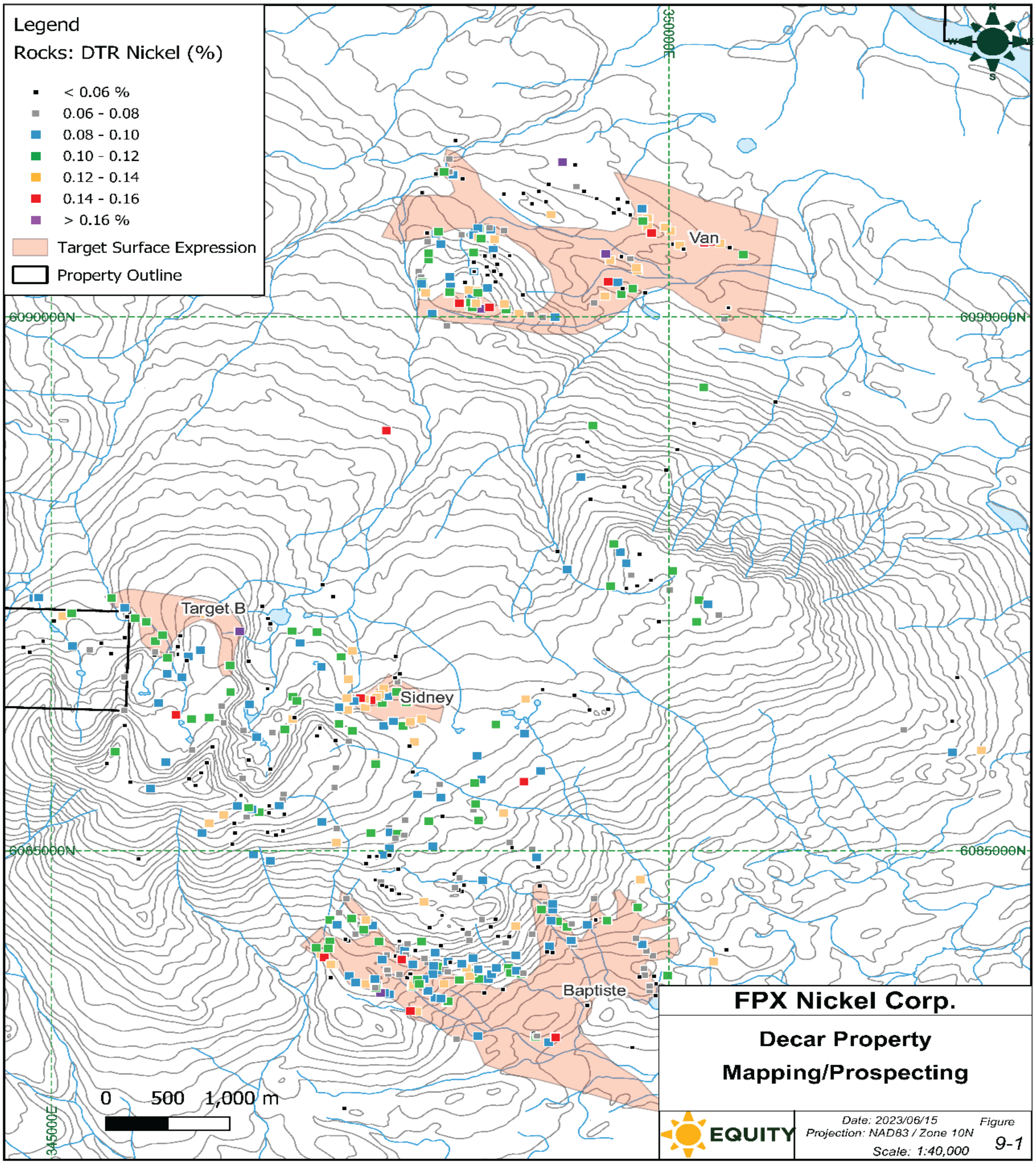
In 2007, FPX conducted a prospecting and rock sampling program over a 4 by 6 km area of the Decar Property that included the Van, B, and Sid targets (Voormeij and Bradshaw, 2008). Sample location coordinates were recorded using a handheld Garmin GPS 60, which typically has a precision of 5-10 m. Samples were mostly serpentinized peridotite (N = 49), of which 42 were analyzed by Acme Analytical Laboratories (ACME) of Vancouver, BC. Total nickel in these samples averaged 0.21% Ni with a range of 0.12% to 0.28%.

Table 9-1: Overview of FPX Surface Samples from the Decar Property

Year	Rock No.	Stream Sediment No.	Petrography No.	Geochemical Assay	Reference
2007	60	N/A	N/A	ACME	Voormeij and Britten, 2008
2008	226	37	13	Portable XRF, some ACME	Britten, 2009; FPX database
2009	130	50	6	Portable XRF, some ACME	Britten, 2010
2010	561	24	N/A	Portable XRF, some ACME	FPX database
2013	35	N/A	N/A	Portable XRF, some Actlabs	FPX database
2014	137	N/A	N/A	Portable XRF, some Actlabs	FPX database
2017	24*	N/A	N/A	Actlabs	FPX database
2021	26	N/A	N/A	Actlabs	Rabb and Carr, 2022
2022	7	N/A	N/A	Actlabs	FPX database
Total	**1,206**	**111**	**19**		

*All 2017 samples were taken from two sumps adjacent to drill holes 17BAP066 and 068.

Figure 9-1: Decar Property Rock Sample Map



Source: Equity Exploration, 2022.

The following year, FPX completed geological mapping, rock sampling, and stream sediment sampling (Britten, 2009). Four surface bulk samples weighing 20 to 120 kg were also collected for metallurgical work. All sample locations were marked with a handheld Garmin GPS 60. Rock samples were collected from outcrop, analyzed with a portable XRF Niton NLp 502 Analyzer (portable XRF) and then slabbed with a diamond saw to help make more accurate visual estimates of awaruite content and grain size. Results showed that awaruite occurs widely across the property, with zones of consistently larger grain size (100-400 µm) defining the Baptiste Deposit and Sid target (Britten, 2009). SEM analysis of 105 awaruite grains from 13 samples indicates an average of 77 wt% Ni and range from 68 to 85 wt% Ni (Le Couteur, 2008).

Seventeen of the 37 sediment samples collected in 2008 were sieved to 80 mesh and then analyzed with the portable XRF; a heavy magnetic fraction was separated from the remaining 20 samples by panning over a strong magnet (Britten, 2009). On average, the heavy magnetic fractions were found to contain 2-4 times more Ni than the 80 mesh sediments.

Geological mapping, rock sampling and stream sediment sampling was expanded in 2009 (Britten, 2010) using methods similar to the 2008 program. 130 rock samples and 50 stream sediment samples were collected. Rock sampling focused mostly on the Van and Baptiste areas, increasing the surface expression of the Baptiste Deposit to a length of 1,750 m and width of 800-1,300 m (Britten, 2010). Three patches of coarse-grained awaruite, separated by overburden, were mapped at the Van target, and could be linked into a single zone up to 700 m in length (Britten, 2010).

Stream sediment samples were dried, sieved to 60 mesh, separated with a pencil magnet, and then analyzed with a portable XRF. These magnetic fractions returned ~1,200-4,800 ppm Ni, with those collected over the Baptiste Deposit consistently in the 3,000-4,800 ppm range and those collected over the Van target between 2,000-3,000 ppm.

The most extensive rock sampling campaign was completed in 2010, with 561 samples collected from the Baptiste, Sid, and B target areas, as well as from elevated ground in the southeastern part of the claim block. Sample descriptions record rock type, serpentinization, magnetic susceptibility, and awaruite grain size. Subsequent rock sampling campaigns focused on the Baptiste (2013), Van (2014) and Sid (2014) areas. This work resulted in the successful delineation of the Baptiste Deposit as a zone of coarse awaruite and also demonstrated significant potential at the Van target, where larger awaruite grains occur over approximately 2.9 km^2 and assays have returned DTR Ni grades similar to Baptiste (FPX, 2018).

The 2017 sampling program involved the collection of chip and sludge samples from sumps located next to drill holes 17BAP065 and 17BAP068. These sump samples returned similar grades to that returned by drilling, with sample grades ranging between 0.03 to 0.13% DTR Ni and averaging 0.09% DTR Ni.

In 2021, FPX completed prospecting and geological mapping within five areas that comprise known awaruite occurrences associated with broader aeromagnetic highs, as well as condemnation sites for planned mine infrastructure (Rabb and Carr, 2022). Outcrops of serpentinized ultramafic rocks were found in two of the five target areas, with the 19 samples collected from these areas returning four samples between 0.06-0.08 wt% DTR Ni. Both comprise new areas of awaruite mineralization. Two of the other target areas were entirely covered by glacial till, vegetation, and/or deadfall, whereas the fifth target area is underlain by listwanite and carbonate-altered peridotite. The condemnation area is underlain by listwanite and semi-massive peridotite.

The 2022 rock sampling program covered ultramafic rocks in the northeastern part of the Property, approximately 6-8 km northeast of the Baptiste Deposit and 5-8 km southeast of the Van target. Three samples of peridotite were collected along with three samples of listwanite and one sample of gabbro dyke. One of the peridotite samples returned 0.06%

DTR Ni whereas all other samples returned <0.01% DTR Ni. Whole rock Ni values range from 1,600-2,300 ppm for peridotite, 300-1,600 ppm for listwanite, and <100 ppm for the gabbro dyke.

9.2 Airborne Surveys

Airborne surveys done on the Decar Property include gradient magnetics and LiDAR (Table 9-2), as summarized in this section.

Table 9-2: Overview of FPX Airborne Surveys

Survey Type	Contractor	Survey Name	Year	Production	Reference
Gradient magnetics	Aeroquest International Ltd	Decar	2010	1639 line-km	Britten and Rabb, 2011
LiDAR	Terra Remote Sensing	Decar	2012	389 km^2	Ronacher, 2013

9.2.1 Gradient magnetics

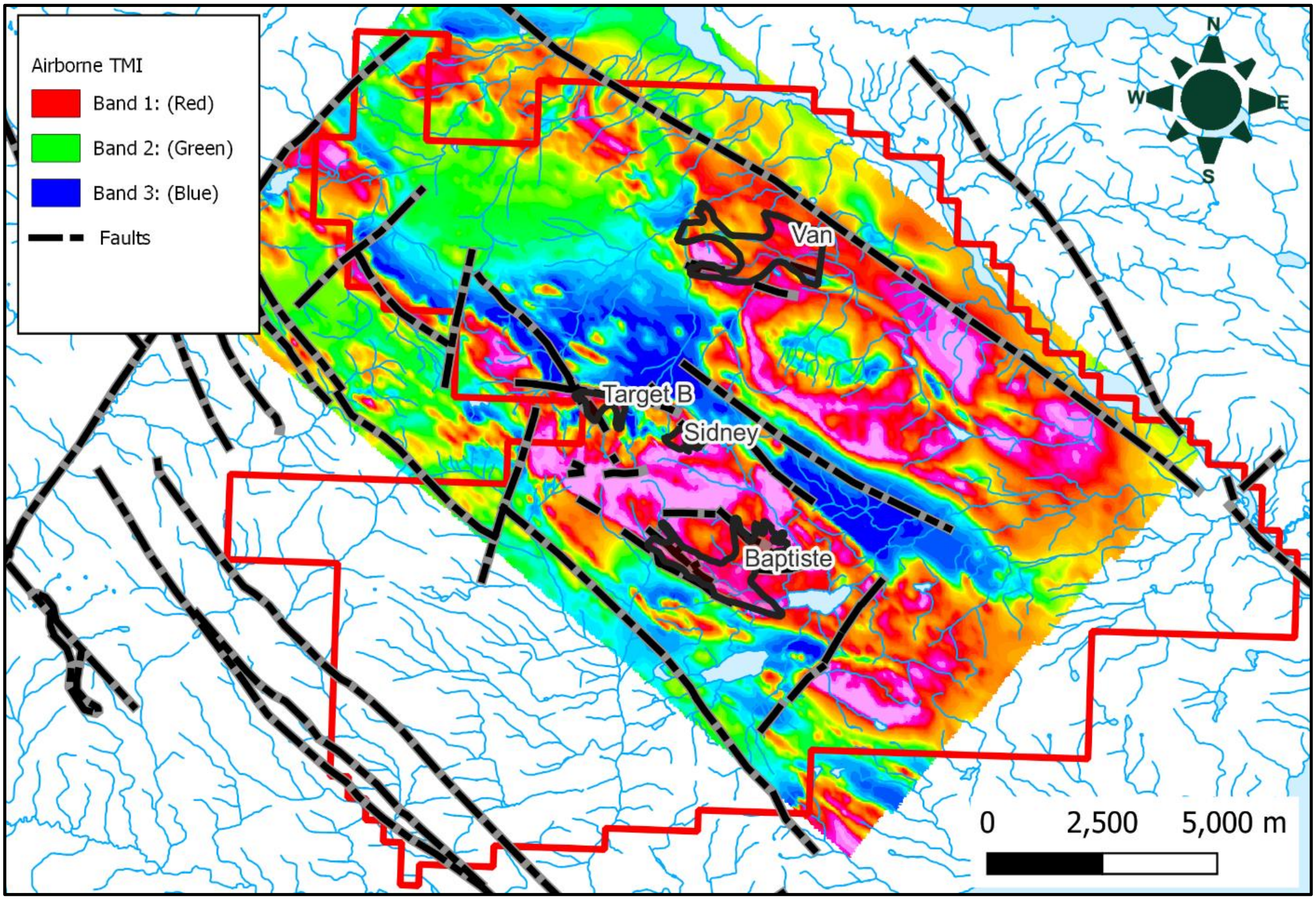
In 2010, FPX contracted Aeroquest International Limited of Mississauga, Ontario, (Aeroquest) to complete an airborne magnetic survey over a portion of the Decar Property. The following description of this program is adapted from Aeroquest's logistical report (Aeroquest, 2010).

The 2010 survey was conducted using Aeroquest's Bluebird Heli-TAG tri-axial magnetic gradiometer, aided by a GPS navigation system, radar altimeter, laser altimeter, orientation sensor, digital video acquisition system, and base station magnetometer (Aeroquest, 2010). Total survey length is 1,668 km, of which 1,639 km fell within the pre-defined project area coordinates, providing coverage over a 220 km^2 area at 150 m line-spacing. Lines were flown at an azimuth of 038°-218°. Nominal tower-bird clearance was between 30-50 m but was periodically higher due to terrain and the capability of the aircraft. The survey speed of 100 km/h and sampling rate of 10 Hz produced a reading approximately every 1.5 to 3.0 m along the flight lines.

Aeroquest delivered six 1:20,000 maps that showed total magnetic intensity (TMI), gradient enhanced TMI with line contours, measured vertical gradient, measured transverse gradient, measured longitudinal gradient and a digital terrain model. All maps were projected in NAD83 UTM Zone 10. No further interpretation was provided by Aeroquest.

Britten and Rabb (2011) used the airborne data to show that much of the Decar Property has a northwest-southeast trending pattern of magnetic highs and lows (Figure 9-2), interpreted as regional-scale strike-slip faults that juxtapose different stratigraphic units of the Cache Creek terrane. They also suggested that the Baptiste, Sid, and Van targets all occur within, or close to the borders of, magnetic highs. A correlation between strong magnetics and awaruite mineralization is consistent with the strongly magnetic nature of awaruite and, more dominantly, associated magnetite.

Figure 9-2: Total Magnetic Intensity (TMI) from the 2010 Airborne Survey



Source: Aeroquest, 2010.

9.2.2 LiDAR

A light detection and ranging (LiDAR) survey was completed over most of the Property in 2012, by Terra Remote Sensing Incorporated of Sidney, British Columbia (Terra Remote). Data processing was completed in fall 2012 and the final report (Terra Remote Sensing Inc., 2013) is publicly filed as an appendix in the 2012 assessment report (Ronacher, 2013). The survey covered a 389^2 area at a line-spacing of 700 m. Terra Remote also took a digital 1:10,000 orthophoto of the Property.

The Lidar survey was flown with a Piper Navajo fixed-wing aircraft based in Burns Lake, BC, approximately 80 km southwest of the Property. The aircraft was equipped with a combined GPS/Inertial Navigation System (GPS/INS) to follow the pre-determined flight lines and maintain a nominal height of 1,150 m above ground level. An average speed of 234 km/h was maintained along with a pulse repetition frequency of 100 kHz and swath speed of 34 times/second, thereby achieving a survey density of 1-2 points/m^2 (Terra Remote Sensing Inc., 2013).

Quality assurance and quality control (QA/QC) of Lidar data was monitored with four control points located within The project area. The difference between known and measured elevations for these control points ranged from +8 cm to -30 cm, and averaged -13 cm (Terra Remote Sensing Inc., 2013). Relative and absolute accuracy is subsequently estimated at ±15 cm and ±20 cm respectively. Lidar points were converted to 1 m contours to provide a more accurate topographic map of the property.

9.3 Ground-Based Geophysical Surveys

Ground-based geophysics done on the Decar Property include IP, resistivity, and magnetics, each of which is described further below.

Table 9-3: Overview of FPX Ground-Based Geophysical Surveys

Survey Type	Contractor	Survey Name	Year	Production	Reference
IP/resistivity and magnetics	P.E. Walcott & Associates	Baptiste grid	2010	20.1 line-km	Britten and Rabb, 2011
		Sidney grid	2010	9.0 line-km	Britten and Rabb, 2011

9.3.1 Ground-based Induced Polarization (IP) and Resistivity

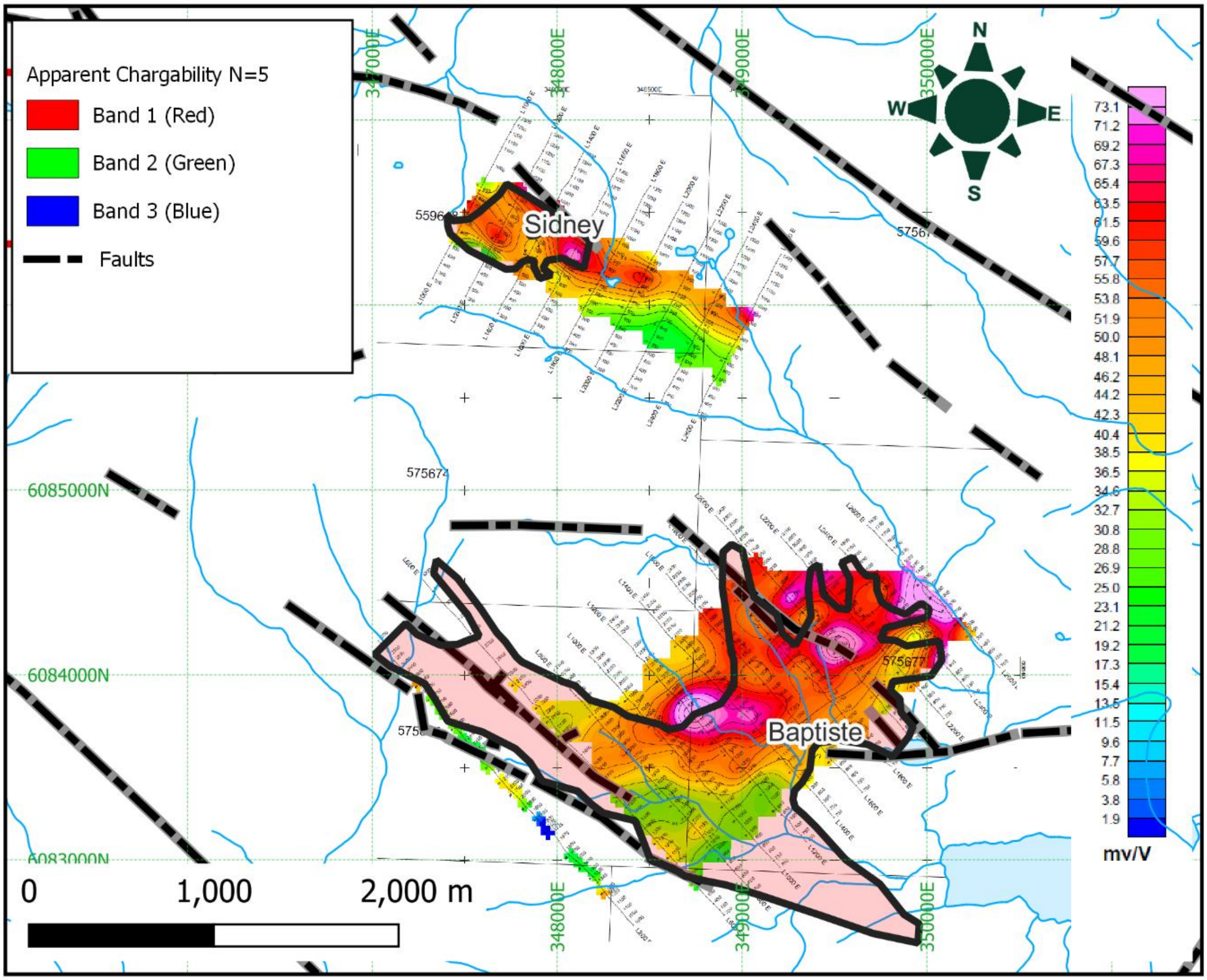
Also in 2010, FPX contracted Peter E. Walcott & Associates Limited of Coquitlam, BC, (Walcott) to conduct a ground-based induced polarization and resistivity (IP) survey on the Decar Property. This work is summarized below.

The 2010 IP survey was designed to delineate regions of increased chargeability after laboratory tests showed awaruite-bearing serpentinized peridotite has elevated chargeability (Walcott, 2011). A total of 29.1 line-km was surveyed over two grids, with 20.1 km on Baptiste and 9.0 km on Sid (Figure 9-3). Survey data was collected with a pulse type system using a pole-dipole array. The transmitter was powered with a 7.5 kW 400 Hz three-phase generator, providing up to 7.5 kW of direct current to the ground. The transmitter cycling rate was set at 2 seconds each for “current on” and “current off”, with the pulses reversing continuously in polarity. Apparent chargeability (mA) is presented as a direct readout of millivolt per volt (mV/V), using a 200 ms delay and a 1,000 ms sample window by the receiver. Locations of stations were acquired using a wide area augmentation system (WAAS) equipped Garmin GPSMAP 60Cx unit.

IP data was presented as individual pseudo-section plots of apparent chargeability and resistivity at 1:10,000 scale. In addition, the third and fifth separation readings were contoured on a plan map at 1:10,000 (Figure 9-3). Ground magnetic data was also contoured at the same scale.

Strong chargeability and resistivity responses were found to correlate with major lithological breaks and/or fault contacts, with no obvious correlation between IP response and awaruite enrichment (Britten and Rabb, 2011).

Figure 9-3: Plan Map of Contoured Chargeability at N = 5 Separations for the Baptiste and Sidney Grids.



Note: Outlines of the Baptiste Deposit and Sidney target, as defined by surface rock sampling are shown as thick black lines.
Source: Walcott, 2011.

9.3.2 Ground-based Magnetic Survey

In 2010, ground magnetic surveys completed as part of the 2010 IP survey showed promise for identifying post-mineral dikes and contacts between serpentinized ultramafic rocks and fault-bounded sedimentary rocks.

In June of 2021, FPX conducted 16 line-km of ground-based magnetometer survey at the Van target to help finalize drill collar locations for maiden exploration drilling (Carr, 2022). Results from this survey confirmed and refined the contacts of two magnetic lows interpreted to be associated with a lithologic contact in the southwest and a zone of incipient serpentinization in the northeast, both considered unfavourable for awaruite mineralization.

9.4 Downhole Surveys

Downhole geophysical surveys were used to characterize various properties of Decar drill core (Table 9-4), including resistivity, natural radioactivity, and magnetic susceptibility. Downhole IP and resistivity testing was done in 2011, whereas magnetic susceptibility testing on drill core was routinely measured as part of all drilling campaigns. Televiewer surveys, which are used to collect high-resolution geotechnical data, are also summarized.

Table 9-4: Overview of FPX Downhole Geophysical Surveys

Survey Type	Contractor	Survey Name	Year	Production	Reference
Downhole IP	Caracle Creek	Vertical profile (VP)	2011	17 drillholes	Palich and Qian, 2012
		X-hole tomography	2011	8 drillhole pairs	Palich and Qian, 2012
		Poly-electric	2011	47 drillholes	Ronacher et al., 2013
		Natural gamma	2011	46 drillholes	Ronacher et al., 2013
Rock Properties	DGI Geoscience	Magnetic IC	2011	42 drillholes	Ronacher et al., 2013
		Focused density	2011	21 drillholes	Ronacher et al., 2013
		Neutron	2011	21 drillholes	Ronacher et al., 2013
Televiewer	DGI Geoscience	Optical	2011	31 drillholes	Ronacher et al., 2013
		Acoustic	2011	36 drillholes	Ronacher et al., 2013

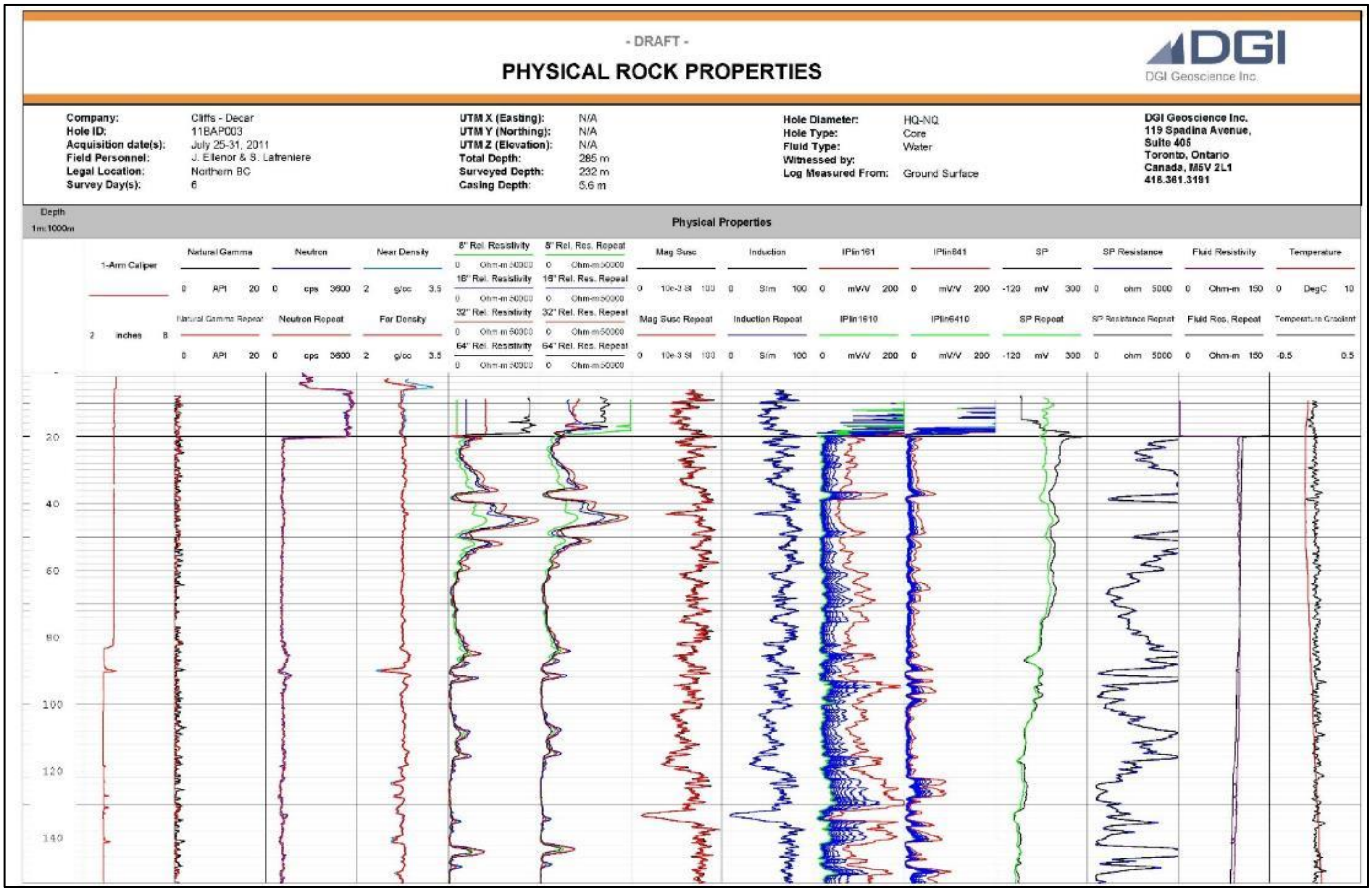
9.4.1 Physical Rock Properties

From 2010 to 2012, Cliffs subcontracted DGI Geoscience Incorporated of Toronto, Ontario, (DGI) to complete downhole physical rock property surveys on all 2010, 2011, and 2012 drillholes. Prior to these surveys, each hole was tested with a dummy probe to see if it was sufficiently open for the physical property probes. Out of the 80 holes drilled from 2010 to 2012, twenty-one were sufficiently open to allow surveys from the top (TOH) to end of hole (EOH), 31 were partially surveyed, and 28 were either obstructed at TOH or not surveyed.

Physical property surveys are used to characterize rock types, define property-specific domains, and constrain geophysical modelling. Probes used for the physical property surveys acquire in-situ data at 10 cm intervals while being lowered and/or raised in the drillhole, with the probes in constant communication with the logging computer at surface. A complete list of holes surveyed as part of this program is given in the previous technical report (Ronacher et al., 2013). DGI delivered the physical property data as databases and strip logs (Figure 9-4).

Poly-electric data was collected in 47 drillholes with a 2PEA-1000 Poly-Electric probe, which measures normal resistivity, fluid resistivity, and fluid temperature (DGI, 2012b) along a vector perpendicular to the drill string. Normal resistivity can mark lithological changes whereas fluid resistivity is needed to correct for the influence of drilling mud and borehole fluid. Changes in fluid temperature can mark zones of water movement.

Figure 9-4: Example of Physical Rock Property Strip Logs Provided by DGI



Note: Strip logs showing (from left to right), borehole diameter, natural gamma, neutron, near density, relative resistivity, relative resistivity repeat, magnetic susceptibility, induction, IP in 161, IP in 641, SP, SP resistance, fluid resistivity and temperature.
Source: DGI, 2011.

A 2PGA-1000 Natural Gamma (γ) probe was used to measure variations in natural radioactivity emitted by uranium, thorium and potassium, and consequently records changes in lithology. In addition, this probe acquires spontaneous potential and single point resistance data that provides additional data on lithology, borehole salinity, and/or formational clay content (DGI, 2012b).

Magnetic susceptibility and inductive conductivity were measured with a MagIC probe and helps delineate lithology by characterizing variance in magnetic mineral content (DGI, 2012b).

9.4.2 Induced Polarization (IP) and Resistivity

Downhole induced polarization and resistivity (DCIP) surveying was completed in September and October 2011 and comprised vertical resistivity and chargeability profiling of 17 boreholes in addition to cross-hole tomographic imagining of eight drillhole pairs (Palich and Qian, 2012). These surveys were completed by Caracle Creek International Consulting Incorporated of Sudbury, Ontario, (Caracle Creek) with the objectives of (1) correlating DCIP and 2010 ground IP surveys,

(2) mapping DCIP anomalies between boreholes, and (3) correlating DCIP features to lithology and awaruite concentrations (Palich and Qian, 2012).

Data was collected with Caracle Creek's proprietary EarthProbe DCIP system. Vertical resistivity and chargeability profiling was achieved by placing a standard current and potential electrode down a single borehole, with measurements taken in time-domain mode using an 8,192 ms current injection square waveform (Palich and Qian, 2012). Based on an electrode separation (a-spacing) of 4 m and 24 electrodes on each cable, the theoretical formation penetration is approximately 25 m off-hole. The EarthProbe survey down 11BAP019 was run with a longer cable (300 m) that allowed for theoretical penetration of 70 m. Cross-hole tomographic surveys were conducted by injecting electrical current between two electrodes across two drillholes, and then measuring the potential difference at the two electrodes immediately below the current injection electrodes (Palich and Qian, 2012). Results provide sections of resistivity and chargeability distribution between the two drillholes that allowed for mapping of conductor continuity.

IP data was presented as resistivity and chargeability strip logs and pseudo sections plotted together with lithology, DTR Ni, Cr, Fe_2O_3 and magnetic susceptibility (Figure 9-5). Results indicate awaruite-bearing peridotite has generally low resistivity (<120 Ω·m) and high chargeability (>25 mV/V) (Palich and Qian, 2012). More resistive zones, with or without chargeability, are broadly associated with unmineralized altered dykes and granitoid rocks.

Cross-hole tomographic data also shows that the more awaruite-enriched peridotite correlates with low resistivity and high chargeability (Palich and Qian, 2012). Low chargeability zones, with or without high resistivity, correlate with lower DTR Ni and typically do not exhibit connectivity between boreholes.

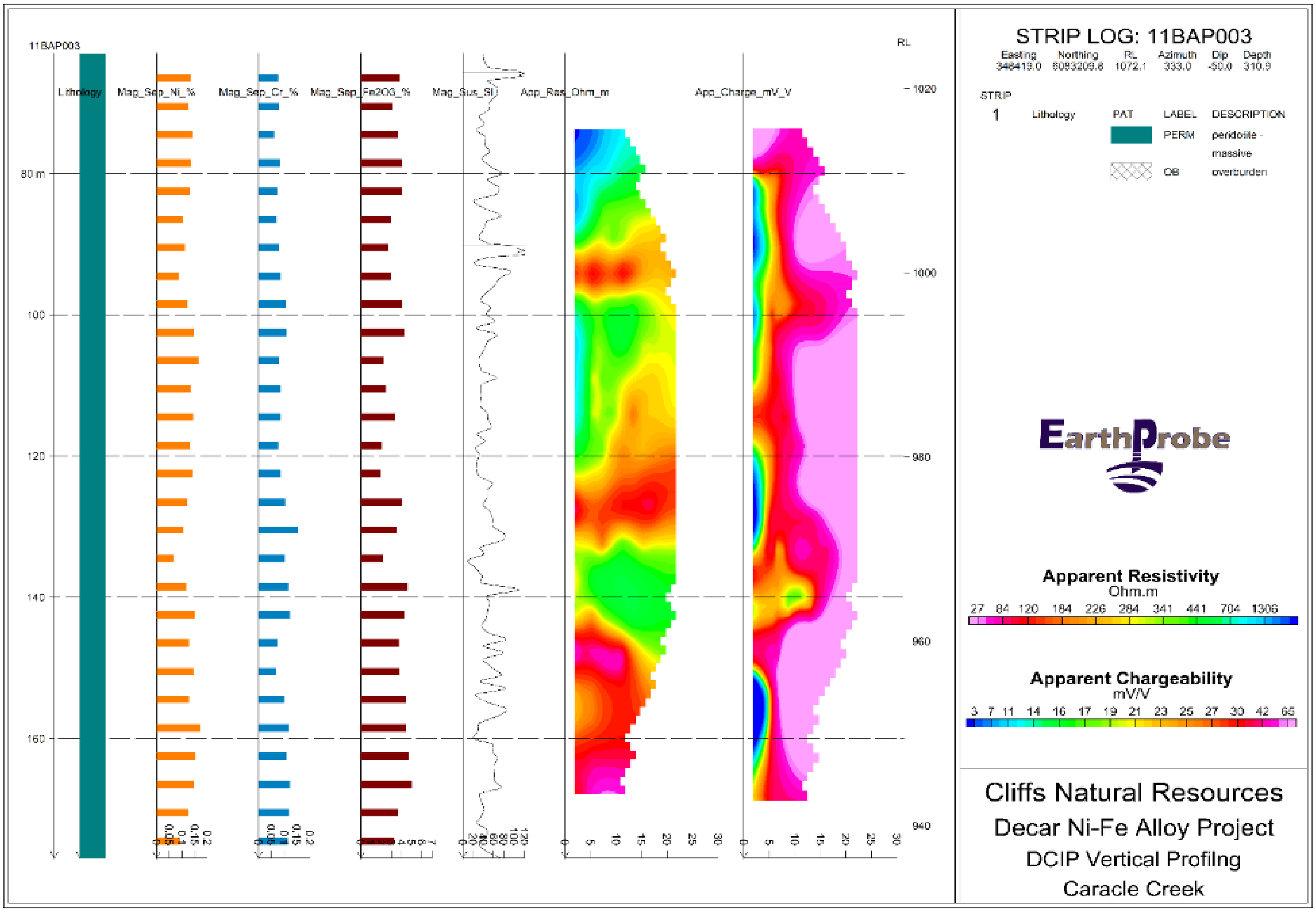
In conclusion, downhole IP surveys indicate that DTR nickel shows a weak positive correlation with chargeability and a weak negative correlation with resistivity (Palich and Qian, 2012). However, Palich and Qian (2012) suggest that this correlation could also reflect magnetite abundance, grain size, and/or multi-element concentration as opposed to just awaruite enrichment, and so suggest caution in using IP data to target awaruite mineralization.

Rock density was measured with a 2GDA Focused Density probe. This probe uses a cesium-137 source to bombard wallrock with intermediate gamma ray energies that are then backscattered and received by the detectors to measure in-situ density.

Neutron porosity was measured with a 2LLP Neutron Probe that uses an alpha emitting radioactive source, americium-241, mixed with beryllium to obtain relative neutron counts. Changes in counts are related to changes in hydrogen ion concentration (DGI, 2012b) that, in turn, may correlate with changes in lithology and/or porosity.

Cluster analysis of physical property data was used to define seven groups, two of which (PP1, PP2) show a strong correlation with high values of DTR nickel (DGI, 2012a). The PP1 group is marked only by low resistivity whereas PP2 is defined by high conductivity and magnetic susceptibility as well as low resistivity, natural gamma, and chargeability. Intermediate DTR Ni grades are associated with low natural gamma and high resistivity (PP5), or high conductivity coupled to low resistivity, chargeability, and natural gamma (PP7).

Figure 9-5: Example of a Downhole IP Strip Log Generated for Drillhole 11BAP003



Note: Drillhole 11BAP003 shows (from left to right) lithology, %Ni in magnetic separate, %Cr in magnetic separate, %Fe2O3 in magnetic separate, magnetic susceptibility, resistivity and chargeability.
Source: Caracle Creek, 2011.

9.4.3 Magnetics

Magnetic susceptibility data was routinely collected as part of all diamond drill programs conducted on the Decar Project. Data was collected by FPX in 2010, Caracle Creek in 2011 and 2012, and Equity Exploration in 2017 and 2021; in all cases using a KT-9 or KT-10 magnetic susceptibility meter. Measurements taken with a KT-9 are not directly comparable to KT-10 but are both reliable for identifying relative magnetic intensities. Samples were taken at 1.0 m intervals in 2010, 0.1 m intervals in the 2011 and 2012 programs, and 0.5 m intervals in the 2017 and 2021 programs.

Strongly magnetic rocks could be defined as having magnetic susceptibility >10 SI units. For the Decar project, 79 of 86 drill holes (92%) have an average magnetic susceptibility that exceeds 10 SI Units. This strong magnetism reflects widespread alteration of olivine to magnetite with or without awaruite, both of which are strongly magnetic. Drill holes with low average magnetic susceptibility typically cut long intervals of metavolcanic and/or metasedimentary country rock or dykes. Weakly mineralized to unmineralized peridotite is also strongly magnetic.

Table 9-5: Overview of FPXs Downhole Magnetic Susceptibility Surveys, with Intensities Measured as SI Units

Year	Holes (N)	%AveDH >10 SI Units	Min AveDH	Median AveDH	Max AveDH
2010	10	100%	27	39	46
2011	16	88%	5	70	96
2012	26	85%	0	75	110
2017	8	100%	40	91	119
2021	26	96%	1	90	132
All	86	92%	0	75	132

9.4.4 Televiewer Surveys

Acoustic and optical televiewer surveys were completed by DGI together with the downhole rock property measurements (see Section 9.2.4). The acoustic televiewer (ATV) probe produces acoustic images of the borehole wall that can be compared with other geological logs to measure the in-situ (or true) orientation of structural features like bedding planes, faults, and fractures (DGI, 2012b). ATV surveys were completed on 17 of the 2011 holes and 19 of the 2012 holes, for a total of 36 surveys.

The optical televiewer (OTV) probe acquires a high-resolution digital image of the borehole wall under in-situ stress, pressure, and temperature conditions (DGI, 2012b), with the aim of identifying and characterizing structural features such as bedding, vein intersections, fractures, and faults. Positional data is measured by an on-board magnetometer and inclinometer sensors, which allow the OTV data to be corrected to true azimuth and dip. OTV surveys were completed on 17 of the 2011 holes and 14 of the 2012 holes, for a total of 31 surveys.

FPX has summarized ATV and OTV data within spreadsheets that record the depth, true dip direction, true dip angle, width (in millimetres), and code for each structural feature. These features include major open, partially open, and minor joint/fractures, bedding/banding/foliation, cleavage, water level, vein, fold, lithology contact, faults, and shear zones. No publicly filed or internal reports are available that summarize the televiewer work although the raw data is well preserved for future use.

10 DRILLING

Diamond drilling for awaruite deposits on the Decar Property was initiated in 2010, with subsequent campaigns in 2011, 2012, 2017, 2021, and 2022, for a total of 40,652 m drilled in 131 holes (Table 10-1). The bulk of these meters were allocated to the Baptiste Deposit and Van target, whereas other drill-tested targets include the B and Sid targets. The following sections provide more details on this drilling.

Drill core sampling, assay methods, and quality assurance/quality control (QA/QC) data is summarized in Section 11.

Composites presented in this section were calculated for intervals with contiguous assays exceeding 0.06% DTR nickel and containing less than 15 m of consecutive samples with grade below 0.06% DTR nickel. When an interval of greater than 15 m of <0.06% DTR nickel was encountered, the composite was split.

Table 10-1: Summary of Drilling Completed by FPX on the Decar Property

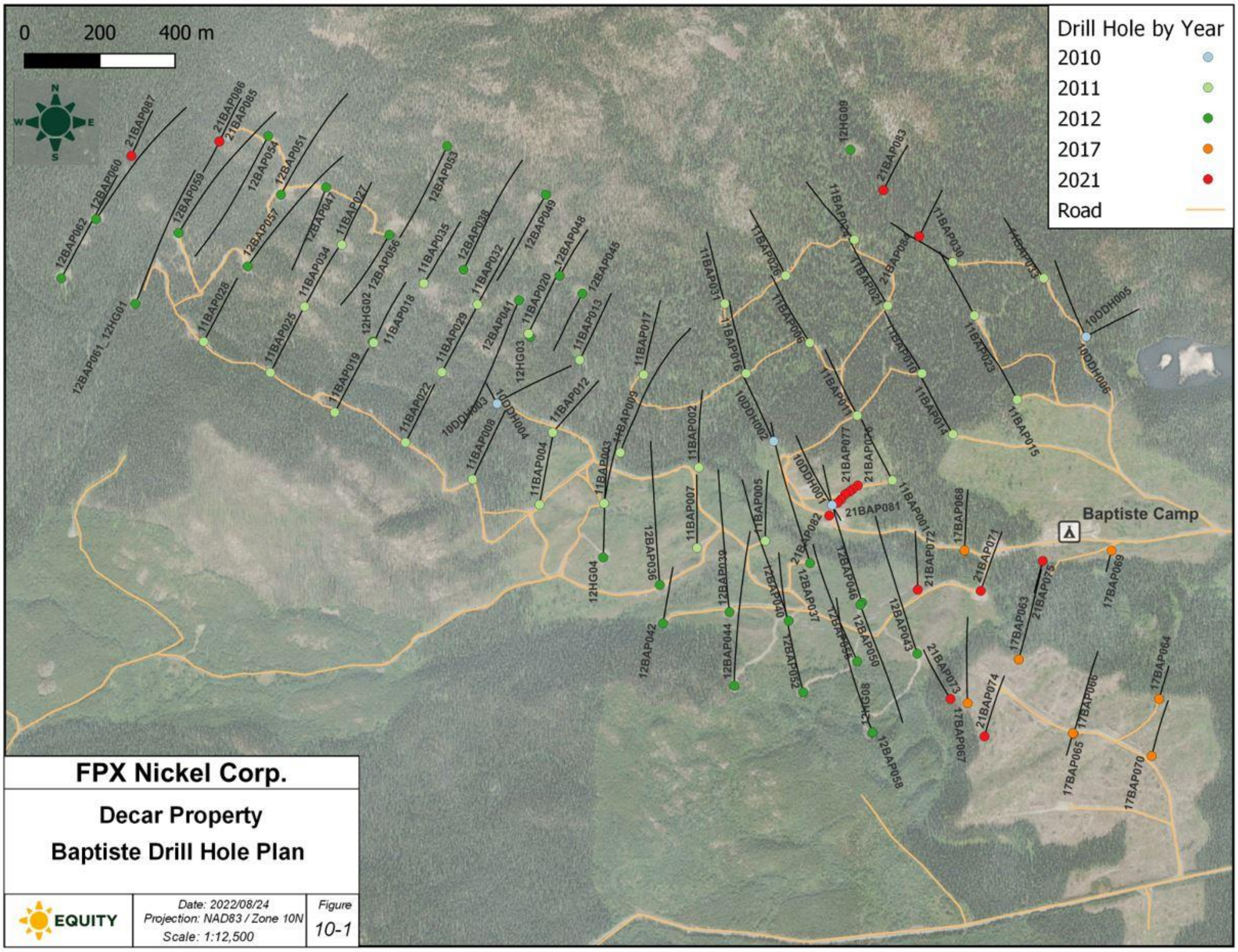
Year	Target	DDH (N)	DDH (Total m)	Avg. m/DDH
2010	Baptiste	7	1,711	244
	Sid	3	847	282
2011	Baptiste	35	10,864	310
	B Target	1	305	305
2012	Baptiste	27	15,096	559
	Baptiste, Engineering	7	1,401	200
2017	Baptiste	8	1,918	240
2021	Baptiste	17	2,856	168
	Van	9	2,688	299
2022	Van	10	2,504	250
	Baptiste, Engineering	7	463	66
Total		**131**	**40,652**	**310**

10.1 Baptiste Deposit

Drilling at the Baptiste site was done as part of all six drilling campaigns on the Decar Property. A summary of collar locations and drill traces at the Baptiste Deposit are presented in Figure 10-1.

Most holes were drilled at dips of -50° to -60° through vertically oriented mineralization, so that the horizontal and vertical extents of the mineralization is equal to 50-65% and 75-85% of their downhole length, respectively.

Figure 10-1: Drill Collar Map for the Baptiste Deposit



Source: Equity Exploration, 2023.

10.1.3 2010 Drilling

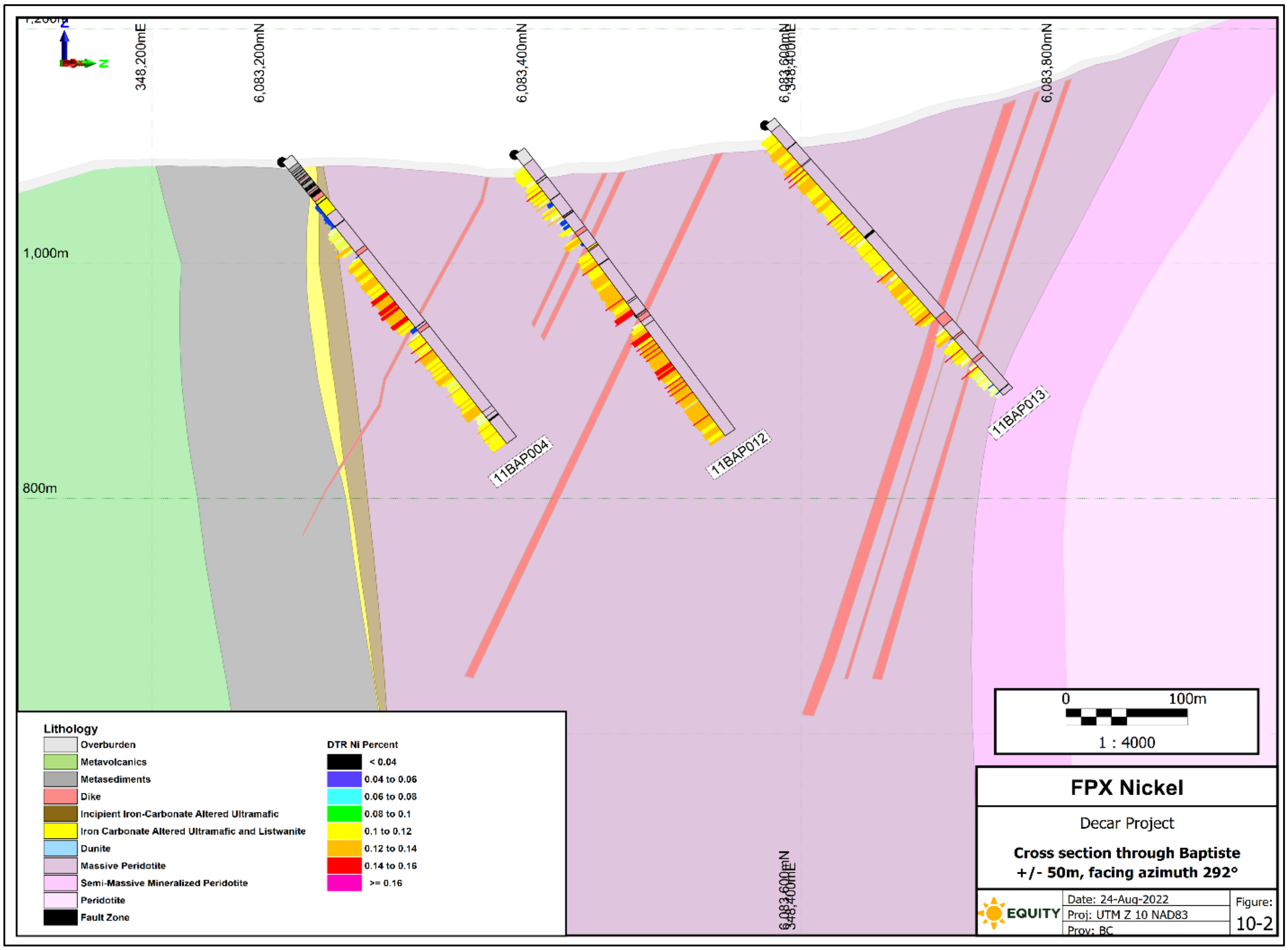
The 2010 drilling program on the Baptiste Deposit was done in July-August and comprised seven holes for 1,711 m. Drilling was completed with a skid-mounted rig provided by Radius Drilling of Prince George, BC.

Holes were spotted using a handheld Garmin GPS 60CSx device with nominal precision of 5-10 m. After completion of the hole, FPX surveyed the collar locations with the same handheld GPS unit. During drilling, boreholes were surveyed by the drill crew using a Reflex single shot instrument. Results show differences of 1.5° to 12.2° between planned azimuths and the upper-most single shot test, suggesting the magnetic nature of the rock may have offset drill alignment, single shot surveys or both. Subsequent downhole testing indicates nominal drill hole deviation rates of -

0.6°/100 m in dip and +1.3°/100 m in azimuth. The consistency in downhole survey data is significantly better than expected for single shot surveys in magnetic rocks.

Most of the 2010 core was drilled as NQ (47.6 mm diameter), except for the top 17.7 m of 10DDH001, which was drilled as HQ (63.5 mm diameter). Drill core was placed in wooden core trays at the drill site that were then labelled with the hole ID and box number and transported to the Decar camp for logging.

Figure 10-2: Section Through the Baptiste Deposit Showing Multiple Drillholes that all end in Mineralization



Source: FPX, 2018.

Each core box was labelled with an aluminum tag indicating the hole number and core interval stored in each box. All 2010 core is currently stored in a fenced compound in Fort St. James, where it is ordered in core storage racks.

Recovery averaged 90.9% and rock quality designation (RQD) ranges from 43-63%, for an average of 50.6%, corresponding to rock quality designation on the boundary separating poor from fair. Magnetic susceptibility was measured at 1 m intervals and very high, with each hole averaging between 27-40 SI units.

Drill core was logged for lithology, awaruite content and magnetic susceptibility. All seven holes reported awaruite mineralization in at least 65% of their hole lengths. One metre of core was sampled for approximately every five metres of core that was drilled.

Assay composites for the 2010 drilling are shown in the table below. Hole 10DDH001 is the central-most of the three holes shown in Table 10-2, with hole 10DDH003 located 740 m west and hole 10DDH006 located 860 m to the northeast. Together with surface data, these three holes suggest contiguous awaruite mineralization over 1.5 km of strike length and to a true vertical depth of 250 m below ground surface. Hole 10DDH007 also intersected high-grade DTR nickel, but the hole was abandoned in a fault zone at 70.1 m depth.

Table 10-2: Composites >20 DTR Ni%*m from 2010 Drilling on the Decar Property

Drillhole ID	From (m)	To (m)	Length (m)	DTR Ni (wt%)	Grade * L	Comments
10DDH001	3.1	322.2	319.1	0.123	39.337	Starts and ends in mineralization
including	3.1	172.5	169.5	0.132	22.291	None
10DDH003	47.2	340.5	293.2	0.138	40.471	Starts and ends in mineralization
10DDH006	13.5	337.0	323.5	0.112	36.172	Starts and ends in mineralization

10.1.4 2011 Drilling

In July to October of 2011, FPX drilled 35 holes into the Baptiste Deposit for 10,864 m. Drilling was completed by three contractors: Apex Diamond Drilling Limited of Smithers, BC, Element Drilling Limited of Winnipeg, Manitoba, and Midpoint Drilling Limited of Langley, BC. These three contractors supplied four drill rigs, two of which were helicopter-portable, one skid-mounted drill rig, and one heli-portable drill rig that was converted to a skid-mounted rig.

The 2011 drilling program was managed by Caracle Creek International Consulting of Sudbury, Ontario (Caracle Creek). Drill collar locations were measured with a differential GPS (DGPS) system. DGPS uses a network of fixed ground-based and GPS satellite systems to achieve nominal accuracy of 10-15 cm.

The drill plan was designed to test the Baptiste Deposit on 13 sections at 200 m spacing, with each section containing one to five drillholes. Section orientation ranges from 028° at the western end of the Deposit area to 332.5° at the eastern end. Each hole from the 2011 program was drilled off its own pad.

Downhole surveys were done with a Reflex Gyro (N = 22) or EZ-shot (N = 13) survey tool. Results from the Gyro survey show nominal deviation rates of -0.1°/100 m for dip and +0.4°/100 m for azimuth. EZ-shot data shows slightly higher deviation rates (-0.5°/100 m for dip; +1.6°/100 m for azimuth) but again, given its similarity to Gyro data, shows that single shot tools performed adequately given the magnetic nature of the bedrock.

The upper-most 100-150 m of each hole was drilled as HQ diam core (63.5 mm diameter) followed by reduction to NQ (47.6 mm diameter) for the remainder of the hole. Procedures for labelling and transporting drill core are the same as those for 2010. All the 2011 core is currently stored in a fenced compound in Fort St. James.

Recovery averaged 91.9% and RQD ranges from ~30-70%, corresponding to rock quality designation of poor to fair. Only 16 of the 35 Baptiste holes have magnetic susceptibility data, with 14 of these averaging a very high 50-100 SI units and two averaging just 5 SI units. The two holes averaging 5 SI units (11BAP025 and 11BAP028) intersected significant intervals of argillite, dike, and non-magnetic carbonate-altered ultramafic.

Drill core was logged for lithology, structure, and awaruite content. One metre of core was sampled for approximately every four metres of core that was drilled.

Table 10-3: Composites >20 DTR Ni%*m from 2011 Drilling on the Decar Property

Drillhole ID	From (m)	To (m)	Length (m)	DTR Ni (wt%)	Grade * L	Comments
11BAP001	86.14	275.2	189.1	0.129	24.3	Ends in mineralization
11BAP003	5.6	310.9	305.3	0.126	38.6	Starts and ends in mineralization
11BAP004	71.0	304.5	233.5	0.126	29.4	Ends in mineralization
11BAP005	45.0	304.5	259.5	0.143	37.1	Starts and ends in mineralization
11BAP007	48.0	304.5	256.5	0.150	38.5	Ends in mineralization
11BAP009	6.0	229.0	223.0	0.131	29.2	Starts in mineralization
	249.0	400.0	151.0	0.134	20.3	
11BAP012	104.5	301.4	196.9	0.141	27.8	Ends in mineralization
11BAP013	9.0	216.0	207.0	0.135	27.9	Starts in mineralization
11BAP014	41.0	265.0	224.0	0.121	27.2	None
11BAP017	17.0	304.9	287.9	0.118	34.1	Starts and ends in mineralization
11BAP018	6.3	301.0	294.8	0.117	34.5	Starts and ends in mineralization
11BAP020	7.0	288.0	281.0	0.122	34.3	Starts and ends in mineralization
11BAP021	21.0	302.0	281.0	0.122	34.2	Starts and ends in mineralization
11BAP022	109.6	301.5	191.9	0.116	22.3	Ends in mineralization
11BAP023	33.0	301.0	268.0	0.127	34.1	Starts and ends in mineralization
11BAP027	118.0	310.5	192.5	0.114	21.9	Ends in mineralization
11BAP029	11.0	298.4	287.4	0.127	36.6	Starts and ends in mineralization
11BAP030	13.0	301.5	288.5	0.127	36.7	Starts and ends in mineralization
11BAP032	6.0	258.0	252.0	0.116	29.2	Starts in mineralization
11BAP034	6.0	298.4	292.4	0.108	31.6	Starts and ends in mineralization
11BAP035	6.0	298.4	292.4	0.112	32.6	None

Assay composites are summarized in Table 10-3. Of the 35 holes drilled in 2011, 21 returned composites that averaged 0.11-0.15% DTR Ni over ~200-300 m of core length for grade x length intervals that exceed 20 DTR Ni% x metre. The remaining 14 holes returned <20 DTR Ni% x metre and can be split into those that show (1) two or more 50-15 m long intervals of mineralization separated by >15 m of weak mineralization (N = 8), (2) one 50-150 m interval of mineralization that starts at 150-250 m core depth and continues to EOH (N = 4), and (3) two to three widely separated intervals of mineralization within a 300 m drill hole, which each interval just 10-50 m thick.

10.1.5 2012 Drilling

From June to October 2012, FPX drilled 27 holes into the Baptiste Deposit for 15,096 m, plus another seven holes, for 1,401 m, to establish hydrogeological monitoring stations and/or test proposed infrastructure sites (Table 10-4). Drilling was done with three drill rigs contracted from Apex Diamond Drilling Limited of Smithers, BC, including two helicopter-portable rigs and one skid-mounted rig.

The 2012 drilling program was again managed by Caracle Creek and followed more-or-less the same procedures as the 2011 program, with the main change being continuous sampling of core as opposed to sampling 1 m for every 5 m or 4 m of drill core as done in the 2010 and 2011 programs, respectively. The 2012 program also included infill sampling of 2010 and 2011 drill core to provide continuous sampling for those holes.

The drill plan was designed to provide infill drilling within the western part of the Baptiste Deposit, as well as expanding it to the west and southeast. Most holes were drilled off their own pad, apart from 12BAP046/050 and 12BAP058/12HG08. Drill collar locations were spotted with a DGPS.

The 27 holes drilled on the Baptiste Deposit have lengths between 300-603 m, with 25 of these ranging from 477-603 m. Hole azimuths range from 028° (or 208°) on the west side of the Deposit to 340° (or 160°) on the east side. Twenty holes were drilled at an inclination of -50°, four were drilled at -60° and three at -65° to -70°. Six of the seven hydrogeological holes were drilled vertically (i.e. -90°), with four of these drilled to 75 m depth and two to 300 m depth. The seventh hydrogeological hole was drilled to 501 m depth at a dip of -50°.

Downhole surveys were done on all non-vertical holes with a Reflex Gyro, with measurements taken at m intervals from the top to the bottom of the hole. Results were similar to the 2010 and 2011 surveys, with nominal deviation rates of -0.2°/100m for dip and +1.3°/100 m for azimuth.

Recovery averaged 94% for all 2012 drilling, whereas RQD ranges from 14% to 76% for individual holes and averages 48%, corresponding to a rock quality designation of very poor to fair.

Table 10-4: Composites >20 DTR Ni%*m from 2012 Drilling on the Decar Property

Drillhole ID	From (m)	To (m)	Length (m)	DTR Ni (wt%)	Grade * L	Comments
12BAP036	31.2	600.2	569.0	0.154	87.8	Starts and ends in mineralization
12BAP037	64.0	216.0	152.0	0.147	22.3	
and	298.0	600.0	302.0	0.146	44.1	Ends in mineralization
12BAP039	38.2	594.1	555.9	0.152	84.5	Ends in mineralization
12BAP040	33.0	588.0	555.0	0.152	84.3	Ends in mineralization
12BAP041	10.0	568.0	558.0	0.136	75.9	Starts in mineralization
12BAP043	33.3	426.0	392.7	0.155	60.9	None
12BAP044	240.0	477.4	237.4	0.154	36.5	None
12BAP045	6.0	487.0	481.0	0.139	66.9	Starts and ends in mineralization
12BAP046	28.5	292.0	263.5	0.142	37.3	Starts in mineralization
and	308.0	494.4	186.4	0.146	27.3	
12BAP047	6.0	600.0	594.0	0.128	75.8	Starts and ends in mineralization
12BAP050	34.5	249.0	214.5	0.140	30.1	Starts in mineralization
12BAP051	182.8	386.0	203.2	0.116	23.7	None
12BAP052	271.0	600.2	329.2	0.154	50.6	Ends in mineralization
12BAP053	334.0	600.0	266.0	0.105	27.9	Ends in mineralization
12BAP054	2.7	600.0	597.4	0.127	75.9	Starts and ends in mineralization
12BAP055	106.0	569.7	463.7	0.156	72.4	Ends in mineralization
12BAP056	5.7	600.0	594.3	0.134	79.7	Starts and ends in mineralization
12BAP057	2.4	554.0	551.7	0.110	60.6	Starts and ends in mineralization
12BAP059	3.8	451.0	447.3	0.136	61.0	Starts in mineralization
12BAP060	156.0	404.0	248.1	0.150	37.1	None
12BAP061	332.0	532.0	200.0	0.120	24.0	None
12HG02	16.0	300.0	284.0	0.131	37.1	Ends in mineralization
12HG03	5.3	300.0	294.7	0.134	39.6	Starts and ends in mineralization
12HG04	176.0	380.0	204.0	0.130	26.6	None

Magnetic susceptibility data for most holes again shows very high average values of 34-110 SI units. A few holes showed noticeably lower average susceptibility (≤5 SI units) that correlate with higher proportions of non-magnetic lithology like Fe-carbonate altered peridotite, listwanite, mafic volcanic, argillite, and dikes.

Drill core was logged for lithology, structure, and awaruite content. Core sampling was done as continuous 4 m long samples from the top to bottom of the hole.

The best intervals from 2012 drilling occur in holes 12BAP036, 12BAP039 and 12BAP040, all of which returned 0.15% DTR Ni over core lengths of 550 to 570 m (Table 10-4). Another 15 holes returned composites averaging 0.11-0.16% DTR Ni over 210 to 600 m of core, with higher grades (0.15-0.16% DTR Ni) occurring over shorter intervals (230-470 m). The eight holes with weak to negligible mineralization were drilled along the northern and southern margins of the Deposit.

10.1.6 2017 Drilling

The 2017 drilling program was comprised of eight holes for 1,918 m, all of which were drilled on the southeastern extension of the Baptiste Deposit. Drilling occurred from August to September 2017 and was done with one drill rig contracted from Apex Diamond Drilling Limited of Smithers, BC.

The 2017 drilling program was managed by Equity Exploration and followed similar procedures set out in the 2011 and 2012 programs. The objective of the drilling program was to expand near surface, high-grade, mineralization at the southeastern end of the Baptiste Deposit. Seven of the eight holes were drilled off their own pad, with holes 17BAP065 and 17BAP066 drilled from the same pad but at azimuths of 014° and 194° respectively. Drill collar locations were initially spotted with a handheld GPS and then surveyed with a real time kinematic (RTK) GPS by HGH Land Surveying from Smithers, BC. This survey also verified the location of some 2011 and 2012 drillholes, as well as the LiDAR base station location.

Six of the holes were drilled from south to north (356° to 014°), with five of these drilled to 250-390 m depth. The sixth hole was stopped at 141 m depth because of the abundant dikes present from the top to bottom of the hole. The two other holes were drilled from north to south (azimuth 194°-195°) to depths of 90-96 m, with the aim of closing off the southwestern margin of the Deposit. All holes were drilled at a dip of -50°.

Downhole surveys were done with the Champ Navigator (Champ Nav) at 5-10 m measurement intervals from the top to the bottom of the hole. Hole deviation rates are similar to previous campaigns, averaging a nominal -0.4°/100 m for dip and +1.1°/100 m for azimuth.

Recovery and RQD measurements are somewhat low relative to previous programs, with recovery averaging 86% and RQD ranging from very poor (17%) to poor (46%), averaging 39% and corresponding to a rock quality designation of poor. Magnetic strength was measured with a KT-10 magnetic susceptibility m and all eight holes show strong magnetic susceptibility (average 40-120 SI units). Those holes with the highest average readings (>80 SI units) intersected long stretches of serpentinized peridotite with higher proportions of magnetite and awaruite. Holes with lower magnetic susceptibility contained more carbonate-altered and hornfelsed peridotite, as well as dikes.

Drill core was logged for lithology, structure and awaruite content. Sampling was done continuously from the top to bottom of the hole.

Table 10-5: Composites >20 DTR Ni%*m from 2017 Drilling on the Decar Property

Drillhole ID	From (m)	To (m)	Length (m)	DTR Ni (wt%)	Grade * L	Comments
17BAP065	29.0	351.0	322.0	0.126	40.6	Starts and ends in mineralization
17BAP067	55.1	348.5	293.4	0.145	42.6	Starts and ends in mineralization
17BAP070	44.0	243.0	199.0	0.100	20.0	None

Assays show that two holes (17BAP065 and 17BAP067) returned intercepts averaging 0.12-0.15% DTR Ni over 290-325 m. Three other holes returned higher grades over shorter intervals (i.e., 0.156% DTR Ni over 117.2 m in 17BAP063) or grades of 0.10-0.13% DTR Ni over 140-200 m of core length. Holes 17BAP064, 17BAP066 and 17BAP069 returned negligible results and so define the southeastern margin of the Baptiste Deposit.

10.1.7 2021 Drilling

The 2021 drilling program was done from June to September and comprised of 17 holes, for 2,856 m. Drilling was focused on further resource definition at the southeastern, northwestern, and northeastern ends of the deposit and was done with two skid-mounted rigs contracted from Apex Diamond Drilling Limited of Smithers, BC. One of these skid-mounted rigs was converted to a fly rig towards the end of the program so that the last five drillholes were supported by helicopter.

Two water crossings were re-established for access into the main camp extending existing logging trails that branch off the 700 road. A series of local bridges providing access for previous campaigns were removed in 2017 as part of a Forest District initiated reclamation work program. The newly constructed bridges were installed by NexTech Forestry Services Ltd. (Nextech).

Out of the 17 holes drilled, 10 were resource definition holes designed to infill and/or expand the Baptiste Deposit and seven were drilled for bulk sample material. The 10 resource holes were drilled at or near the southeastern (21BAP071 to 21BAP075), northeastern (21BAP083, 21BAP084), and northwestern (21BAP085 to 21BAP087) extent of the Baptiste Deposit. These 10 holes were all either in gaps between previous drill holes or aimed to expand on historical results. The 10 resource definition holes were mostly at azimuths of 015°-030° or 195° to 201°, dip of -60°, and to depths of between 175 to 350 m. These 10 holes were drilled off a total of nine pads. The remaining 7 holes were to help delineate a potential future bulk sample location, and these holes were all subvertical and drilled to approximately 21 m in length.

Drill holes were spotted using a handheld GPS and final collar coordinates were later surveyed using a differential global positioning system (DGPS). Drill rig alignment was done using a north seeking gyro for azimuth and dip directions. Final downhole surveys were conducted for each hole using the same north seeking gyro equipment. Holes were surveyed at 3 m increments and show deviation rates similar to previous campaigns, averaging -0.5° for dip and +0.9°/100 m for azimuth.

Drill holes were started as HQ (63.5 mm diameter) for the first 100 to 150 m and were reduced to NQ (47.6 mm diameter) for the remainder of the hole. This was done to prevent losing drill holes in unstable ground conditions, allowing for flexibility to reduce if required.

Drill core was transported from the rig to the core logging facility using 4x4 truck and helicopter. Core boxes were labelled with hole number, box number and mage. Wet and dry photographs were taken of all drill core. The core was logged for geology (lithology, alteration, mineralization, veining, and structure), core recovery, RQD, magnetic susceptibility (using KT-10 handheld probe) and point load testing. Whole core samples were taken at approximately 30 m intervals for uniaxial compression testing (UCS).

Recovery and RQD measurements are higher than the 2017 program and similar to the 2010-2012 work, with recovery for the 10 Baptiste holes averaging 93% and RQD ranging from poor (30%) to fair (74%) and averaging 48%. Magnetic strength was measured with a KT-10 magnetic susceptibility m, with all 17 holes showing strong magnetic susceptibility (average 50-120 SI units). Those holes with the highest average readings (>100 SI units) generally intersected longer stretches of cataclastic serpentinized peridotite with higher proportions of magnetite and awaruite.

Drill core selected for sampling was cut in half using core saws, collected and sent to Actlabs of Kamloops, BC for analysis. The five holes drilled near the southeastern extent of the Deposit include four that returned 0.12-0.15% DTR nickel over core lengths of 190-320 m and a fifth hole that assayed 0.12% DTR nickel over 137 m and ended in mineralization (Table

10-6). The two holes drilled at the northeastern extent averaged 0.10-0.11% DTR Ni over 180-185 m of core length, with 21BAP084 ending in weak mineralization. The three holes drilled at the northwestern extent of the deposit returned 0.10-0.12% DTR Ni over core intervals of 100-260 m with two of these holes ending in mineralization.

The seven holes drilled to delineate a potential future bulk sample averaged between 0.08 to 0.11% DTR Ni from the top to bottom of each hole, which in this case ranged from 18-21 m. Several of these holes collared in unmineralized rocks so that removing these samples returns more strongly mineralized intervals of 0.10-0.15% DTR Ni over 8.0-20.0 m of core length.

Table 10-6: Composites >20 DTR Ni%*m from 2021 Drilling on the Baptiste Deposit

Drillhole ID	From (m)	To (m)	Length (m)	DTR Ni (wt%)	Grade * L	Comments
21BAP071	47.0	241.0	194.0	0.119	23.1	Ends in mineralization
21BAP072	39.0	285.0	246.0	0.139	34.1	Ends in mineralization
21BAP073	48.1	303.0	254.9	0.151	38.4	Ends in mineralization
21BAP074	33.8	351.0	317.2	0.135	42.9	Ends in mineralization
21BAP075	39.0	175.6	136.6	0.120	16.5	Ends in mineralization
21BAP083	18.0	198.0	180.0	0.111	19.9	None
21BAP084	18.0	201.0	183.0	0.099	18.1	Ends in weak mineralization
21BAP085	77.9	180.0	102.2	0.115	11.7	Ends in mineralization
21BAP086	3.2	264.5	261.3	0.101	26.4	Ends in mineralization
21BAP087	3.0	243.0	240.0	0.122	29.4	None

10.1.8 2022 Drilling

In 2022, seven vertical holes between 35 to 93 m in length, were drilled at the Baptiste site (but outside of the Baptiste Deposit area) for purposes of collecting geotechnical data. All these holes were logged and two of them returned awaruite mineralization, although neither returned more than 20 DTR Ni%*m owing to the short length of the holes.

10.2 Van Target

The Van target lies 7.4 km north-northeast of the Baptiste Deposit, near the northeastern margin of the Trembleur ultramafite (Figure 7-3). This target was defined by geological mapping and rock sampling (Figure 9-1) and was first drilled in 2021 with a program consisting of nine holes for 2,688 m. This was then followed up in 2022 with a program consisting of ten holes for 2,504 m (Table 10-1, Figure 10-3).

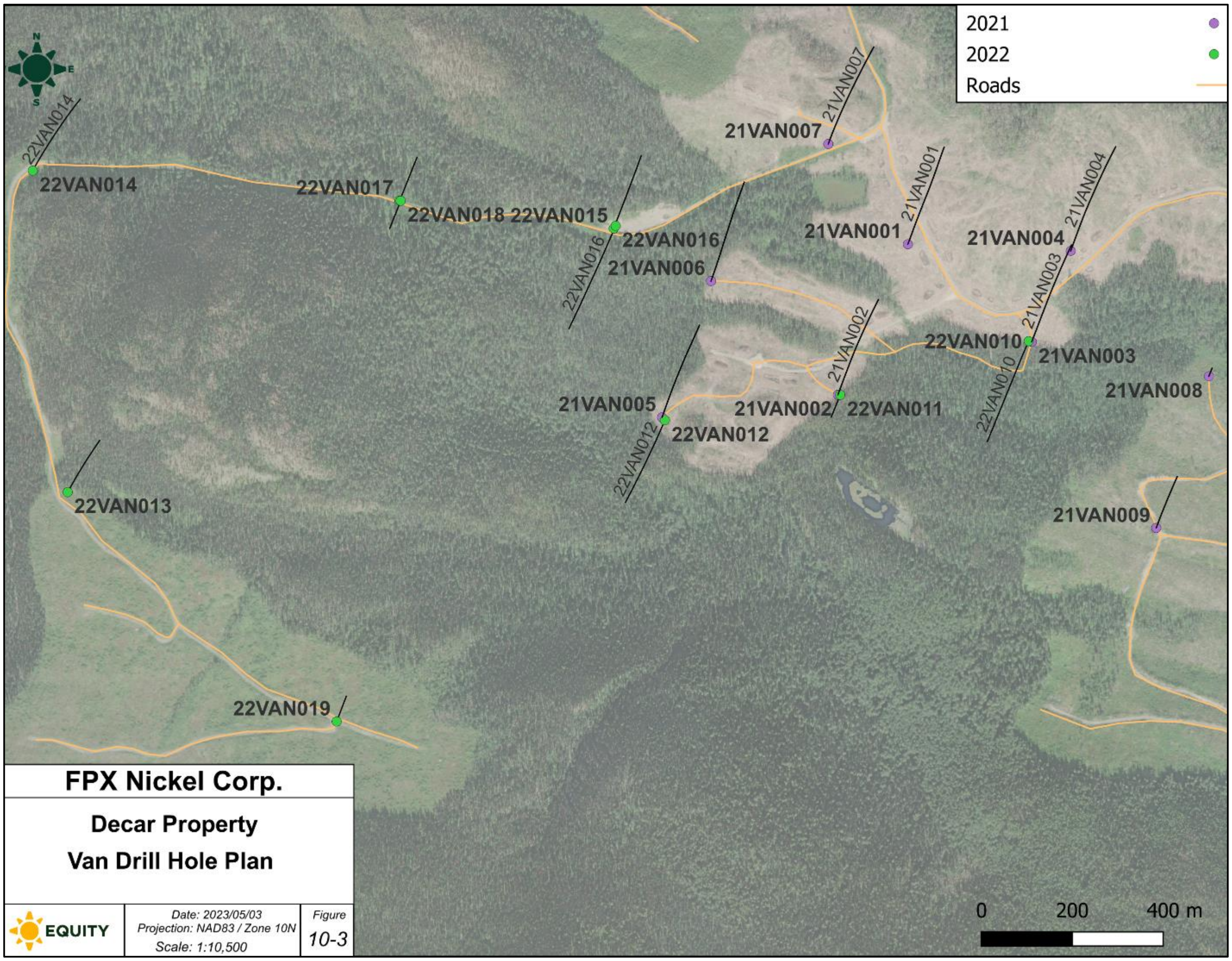
10.2.1 2021 Drilling

The 2021 drill program consisted of nine holes, seven of which were drilled to between 290-350 m depth, whereas 21VAN009 was drilled to just 186 m and 21VAN008 was abandoned at 30 m depth. The drill subcontractor and methods were the same as those used for the 2021 drilling at the Baptiste Deposit.

Drilling was done with one of the skid-mounted rigs contracted from Apex and was completed on north-northeast trending sections. Starting azimuths for all drill holes were 020° and starting dips were -50°. Drill holes were drilled as HQ (63. mm diameter).

Drill holes were spotted using a handheld GPS and final collar coordinates were later surveyed using a differential GPS (DGPS). Drill rig alignment was done using a north seeking gyro for azimuth and dip directions. Final downhole surveys were conducted for each hole using the same north seeking gyro equipment and show deviation rates of +0.9°/100 m for dip and +1.4°/100 m for azimuth. The plan map showing all holes drilled for the Van target are in Figure 10-3 below.

Figure 10-3: Plan Map for the Van Target Showing Holes Drilled in 2021 and 2022

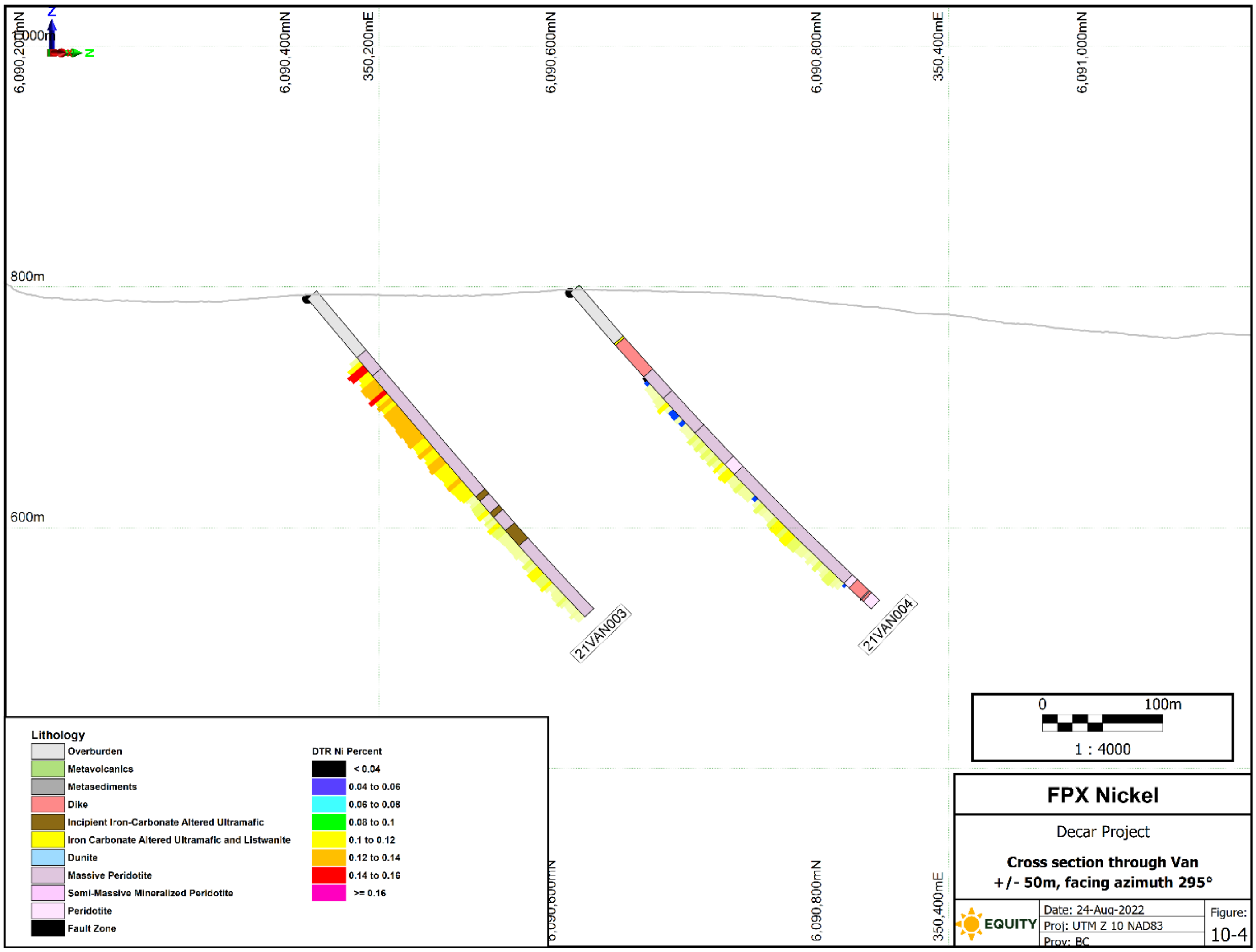


Source: Equity Exploration, 2023.

Drill core was transported from the rig to the core logging facility with a 4x4 truck. Core boxes were labelled with hole number, box number and mage. Wet and dry photographs were taken of all drill core. The core was logged for geology (lithology, alteration, mineralization, veining, and structure), core recovery, RQD, magnetic susceptibility (using KT-10 handheld probe), and point load testing. Core sampling was done as four-metre-long continuous samples from the top to the bottom of the hole.

Recovery and RQD measurements are generally higher than the Baptiste Deposit, averaging 97% and 78%, respectively. RQD is notably better, ranging from fair to excellent for all nine of the holes, with an average of good (78%). Magnetic susceptibility measurements show that seven of nine holes have very strong susceptibility, averaging between 80-130 SI units, while two holes have average susceptibility ≤15 SI units. The two less magnetic holes (21VAN008, 21VAN009) intersected significant intervals of non-magnetic carbonate-altered peridotite, listwanite, altered dike, and/or argillite.

Figure 10-4: Section Through the Van Target Showing Two Drillholes that End in Mineralization



Source: Equity Exploration, 2022.

Table 10-7: Composites > 20 DTR Ni%*m from 2021 Drilling on the Van Target

Drillhole ID	From (m)	To (m)	Length (m)	DTR Ni (wt%)	Grade *L	Comments
21VAN001	34.5	208.0	173.5	0.129	22.3	Another 114 m interval starting at 240 m ends in mineralization
21VAN002	24.1	351.0	326.9	0.123	40.2	Ends in mineralization
21VAN003	64.0	351.0	287.0	0.119	34.2	Ends in mineralization
21VAN004	101.0	332.0	231.0	0.090	20.7	None
21VAN005	9.0	351.0	342.0	0.128	43.9	Ends in mineralization
21VAN006	0.0	216.0	216.0	0.117	25.2	None

Drill core selected for sampling was cut in half using core saws, collected, and sent to Actlabs of Kamloops, BC for analysis. The first six holes of the program all returned at least one interval with 0.09-0.13% DTR nickel over 170-340 m of core length, with four of these holes ending in mineralization (Table 10-7, Figure 10-4). Hole 21VAN007 returned a long interval of low-grade mineralization that could possibly mark the northwestern extent of the Van target. Holes 21VAN008 and 21VAN009 returned relatively high proportions of unmineralized lithologies and appear to mark the eastern end of the target. The southern, northern, and western extents of the target were still open after the 2021 drilling.

10.2.2 2022 Drilling

The 2022 program consisted of ten holes, four of which were returned significant mineralization and were drilled to depths of 316 to 439 m and five that were stopped between 106 m and 298 m due to lack of mineralization. One hole, 22VAN011, was abandoned at 90 m due to bad ground, but the mineralization looked prospective for the meterage drilled. The drill contractor for 2022 was Foraco Canada of Kamloops, BC, and methods were generally the same as those used for the 2021 drilling at the Van target.

Drilling was done with a skid-mounted rig and was completed on north-northeast trending sections. Starting azimuths for drill holes were either 022° or 203° and starting dips were -55°. Drill holes were drilled HQ (63.5 mm diameter) except for holes 22VAN010, 22VAN011, 22VAN012, and 22VAN015 which were drilled to bedrock with HQ and then reduced to NQ (47.6 mm diameter) for the rest of the hole.

Drill holes were spotted using a handheld GPS and final collar coordinates were later surveyed using a differential GPS (DGPS). Drill rig alignment was done using a north seeking gyro for azimuth and dip directions. Final downhole surveys were conducted for each hole using the same north seeking gyro equipment used in 2021.

Drill core was transported from the rig to the core logging facility with a 4x4 truck. Core boxes were labelled with hole number, box number and mage. Wet and dry photographs were taken of all drill core. The core was logged for geology (lithology, alteration, mineralization, veining, and structure), core recovery, RQD, magnetic susceptibility (using KT-10 handheld probe) and point load testing. Core sampling was done as four-metre-long continuous samples from the top to the bottom of the hole.

Recovery and RQD measurements are slightly lower than the 2021 results, averaging 94% and 77% respectively. The RQD ranges from fair to excellent across the 10 holes with an average of good. Magnetic susceptibility measurements are high for all 10 holes, averaging between 25 and 87 SI units for each hole within a range that extends from 0 to at least 120 SI units. Some of the less magnetic holes contain up to 20 m of carbonate-altered peridotite, whereas others consist only of peridotite.

Drill core selected for sampling was cut in half using core saws, collected, and sent to Actlabs of Kamloops, BC for analysis. Two holes from this program returned significant intervals of mineralization, including 346 m with 0.133% DTR nickel in 22VAN010 and 426 m with 0.133% DTR nickel from 22VAN016. Both holes also end in mineralization. Hole 22VAN012 returned an interval of 172 m with 0.143% DTR nickel, showing decreasing mineralization with depth that possibly marks the southwest limit of the target (Table 10-8). Hole 22VAN015 and 21VAN007 returned comparable long intervals of low-grade mineralization that could possibly mark the northwestern extent of the Van target. Holes 22VAN018 and 22VAN019 are just below the threshold of 20 DTR Ni%*m. Holes 22VAN013, 22VAN014 and 22VAN019 returned long intervals of unmineralized ultramafic rock and so appear to close off the western and southwest ends of the Van Target. The southern extent of the target is still open.

Table 10-8: Composites >20 DTR Ni%*m from 2022 Drilling on the Van Target

Drillhole ID	From (m)	To (m)	Length (m)	DTR Ni (wt%)	Grade*L	Comments
22VAN010	51.26	397.60	346.34	0.133	46.1	Ends in mineralization.
22VAN012	6.00	178.00	172.00	0.143	24.6	178 to EOH is not well mineralized.
22VAN016	12.40	439.30	426.90	0.126	53.8	Ends in mineralization.

10.3 B Target

The B target is located 4.8 km northwest of the Baptiste Deposit and, in 2011, was tested with a single hole for 305 metres. The subcontractors and methods involved in this drill program are the same as those for the 2011 Baptiste drilling.

Downhole surveys were done with a Reflex single shot tool and show relatively high deviation for both dip (-0.9°/100 m) and azimuth (-3.7°/100 m). Approximately 88% of the hole length was recovered as drill core and RQD averaged 54% for the length of the hole, which is similar to the Baptiste Deposit.

Drill core was logged for the same parameters as the Baptiste holes, with the B target hole comprising mostly peridotite with minor amounts of volcanic rock and listwanite.

A composite calculated from 1 m samples taken every 4 m returned 268.5 m of 0.14% DTR nickel from 32.5 m to 301.0 m depth, equivalent to an above-average intercept at Baptiste Deposit.

10.4 Sid Target

The Sid target is located 3.1 km north-northwest of the Baptiste Deposit and, in 2010, was tested with three diamond drill holes. The subcontractors and methods involved in this drill program are the same as those for the 2010 Baptiste drilling, with Radius converting one of their skid-mounted rigs to a fly rig. Three holes were drilled for a total of 847 m, with one of these holes abandoned at 103 m depth.

Downhole surveys as measured with a Reflex single shot instrument showed an average dip deviation of -0.4°/100 m and azimuth of 1.8°/100 m, which is broadly similar to the deviation recorded in the Baptiste Deposit. Holes are currently stored in a fenced compound in Fort St. James.

Drill core lithology consists mostly of serpentinized peridotite but also contains significant intervals of massive unmineralized peridotite, carbonate-altered peridotite, listwanite, and/or fault zone. All three holes host long intervals of awaruite mineralization, totalling 299 m in 10DDH009 (87% of all core) and 321 m in 10DDH010 (81%). The average grain size of awaruite is described as similar to the Baptiste Deposit.

Drill core was sampled as 1 m samples every fifth metre along the drill core and was cut in half using core saws, collected, and sent to ACME of Vancouver, BC for analysis. Composites calculated from this sporadic but systematic sampling are shown in Table 10-9.

Table 10-9: Composites >20 DTR Ni%*m from 2010 Drilling on the Sid target

Drillhole ID	From (m)	To (m)	Length (m)	DTR Ni (wt%)	Grade * L	Comments
10DDDH009	127.75	345.95	218.2	0.11	23.5	1 m out of every 5 m sampled, ends in mineralization
10DDDH010	109	398	289	0.13	38.6	1 m out of every 5 m sampled, ends in mineralization

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sampling Methods

Drill core sampling for the Decar Property was completed in seven campaigns, with six of these done concurrent to drilling and one comprising an infill sampling program of 2010 and 2011 drill core in 2012. Collectively, these campaigns have sampled and analyzed 90% of the core drilled by FPX on the Decar Property. The 10% of unsampled core consists mostly of non-ultramafic rock types and the three selectively sampled holes drilled on the Sid target that were skipped in 2012 infill sampling campaign.

On-site sample preparation procedures were similar for all sampling campaigns. In each case, the selected drill core intervals were cut with a core saw, with one half placed into a plastic bag with a pre-numbered sample tag (core sample) and the other half placed back in the core box for reference (reference core). Standards (CRM, non-certified standards, replicates) and blank samples were alternately inserted as every 10th sample for a total insertion rate of approximately 10%. One core duplicate was also inserted in every 20 samples for the 2010, 2011, 2012, and 2017 programs but were reduced to one in every 40 samples for the 2021 and 2022 work. Core duplicates comprise a quarter core sample obtained by splitting the reference core, with the core duplicate then placed in its own plastic bag with a pre-numbered sample tag. Coarse core preparation duplicates were also inserted at a rate of 1 in every 20 samples for the 2010, 2011, 2012, and 2017 programs but were dropped from the 2021 and 2022 work.

Initial sampling of 2010 core was conducted as 1 m samples every fifth metre along the drill core that meant only 18% of core was sampled concurrent with 2010 drilling (Table 11-1).

Core sampling concurrent with the 2011 drill program involved the collection of 1 m samples every fourth metre along the drill core, irrespective of rock type (Ronacher, et al., 2013). This sampling method was designed to preserve core for potential follow-up metallurgical testing and equated to sampling of 23% of all metres drilled.

Table 11-1: Overview of Decar Drill Core Sampling Campaigns

Drill Campaign	Sampling Campaign	Quantity	Core Samples Assayed		% of Length
			Total Length (m)	Average L (m)	
2010	2010 drilling	454	466.9	1.0	18%
	2012 resampling	345	1,272.4	3.7	50%
2011	2011 drilling	2,620	2,618.8	1.0	23%
	2012 resampling	2,600	7,578.7	2.9	68%
2012	2012 drilling	4,192	15,552.3	3.7	94%
2017	2017 drilling	460	1,565.1	3.4	82%
2021	2021 drilling	1,384	4,842.1	3.5	87%
2022	2022 drilling	657	2,346.7	3.6	79%
Total		**12,712**	**36,243.0**	**2.9**	**89%**

In 2012, sampling was done as contiguous 4 m intervals rather than regularly spaced intervals, so that 94% of the 2012 core had been sampled by completion of the drilling program.

Infill sampling of 2010 and 2011 core was also completed in 2012 to close the 3-4 m sampling gaps between the regularly spaced 1 m samples taken concurrent with those programs. As a result, the total metres sampled from these drill holes increased from 18% to 68% for the 2010 core and from 23% to 91% for 2011 core.

The 2017, 2021, and 2022 core sampling program were all done in contiguous 3 to 4 m intervals so that between 79% to 87% of all drill core had been sampled by the completion of the program. Differences in the proportion of metres sampled between these campaigns relates to the thickness of overburden and/or the number of geotechnical holes, which are typically less thoroughly sampled. For example, out of the seven geotechnical holes drilled on the Baptiste Deposit in 2022, only two were sampled.

11.2 Sample Shipping and Security

For all sampling campaigns, core samples were sealed in plastic bags with a zap strap, then aggregated into groups of 5-10 samples that were sealed into rice bags with a uniquely numbered security tag. These rice bags were then transported to shipping depots using a locally contracted expediting service.

In 2010 and 2011 programs, rice bags were shipped from the Decar property to Smithers, BC, through CJL Enterprises Ltd of Smithers, BC. From there, samples were shipped to Acme Analytical Laboratories Ltd in Vancouver, BC, (2010) and Actlabs facility in Ancaster, Ontario, (2011) by Bandstra Transportation Systems Ltd (Bandstra).

In 2012, sample transport from the Decar property to Smithers, BC, was handled by Rugged Edge Holdings Ltd of Smithers, BC, and Bandstra then transported these samples to the Actlabs facility in Kamloops, BC, for analysis.

In 2017, core samples were transported to Prince George, BC, by Equity Exploration and then to Actlabs in Kamloops, BC, by Bandstra.

For the 2021 and 2022 programs, core samples were transported from the Decar property to Prince George, BC, by Newlands Enterprises Ltd of Fort St. James, BC, and then to Actlabs in Kamloops, BC, by Bandstra.

All drill core for the Decar property has been transported to Fort St. James, BC, where it is stored in a fenced compound owned by Russell Transfer Ltd. Core boxes are stored in racks and are generally well preserved as of the June 2022 site visit (see Section 12).

11.3 Analytical Methods

Geochemical assay of the core samples collected during 2010 drilling were initially completed by ACME Analytical Laboratories Ltd of Vancouver, BC (ACME), an ISO 9001 accredited laboratory that was purchased by Bureau Veritas Mineral Laboratories in 2012.

All subsequent analyses for the Decar property were completed by Activation Laboratories Limited of Kamloops, BC, and Ancaster, Ontario (Actlabs). This includes the 2011 reanalysis of coarse rejects from the seven Baptiste holes drilled in 2010. The three 2010 holes from the Sid target were not re-analyzed. Both Actlabs facilities are independent commercial laboratories that are ISO/IEC 17025:2005 accredited, meaning they meet the general competency requirements to carry out tests and/or calibrations using standard, non-standard, and laboratory-derived methods (ISO, 2005).

All core samples have been analyzed twice, with one analysis on a whole rock pulp and another on a magnetic separate generated with a Davis Tube magnetic separator. The initial preparation stage is the same, and the entire core sample crushed, then split into a 250 g sub-sample pulverized to a pulp with 95% of particles passing 75µm (200 mesh).

For whole rock analyses, a portion of the sub-sample is fused with lithium metaborate/tetraborate flux and analyzed by inductively coupled plasma optical emission spectrometry (ICP-OES). These analyses are here abbreviated as fusion/ICP.

The magnetic separate sample is generated by running the pulverized sub-sample through a Davis Tube magnetic separator, which splits the sub-sample into magnetic and non-magnetic fractions. The magnetic fraction is then fused with lithium metaborate/tetraborate flux and analyzed by X-ray fluorescence (XRF), here abbreviated as fusion/XRF. These analyses are more representative of the minable grade of the deposit since most recoverable nickel will be in the magnetic separate (e.g., awaruite) whereas the whole rock fusion/ICP analyses may include non-recoverable nickel hosted in silicate phases like olivine.

Table 11-2: Overview of Analytical Methods Used by FPX for Drill Core

Campaign	Lab	Preparation		Fusion/ICP		Magnetic Separation		Fusion/XRF	
		Code	Method	Code	Method	Code	Method	Code	Method
2010 drilling	ACME	R200-250	250 g passing 75 µm	1E	4-acid digest, ICP	n/a	n/a	8FPX	"Metallic Ni by FPX method"
2011 drilling	Actlabs	RX-1SD	250 g passing 75 µm	4B	Lithium borate fusion, ICP-OES	8-DTMS	Davis Tube magnetic separation	4C	Lithium borate fusion, XRF
2011 re-assay									
2012 drilling									
2012 infill									
2017 drilling									
2021 drilling									
2022 drilling									

11.4 Quality Assurance and Quality Control (QA/QC)

QA/QC of geochemical analyses was monitored with certified reference materials (CRM), non-certified standards (NCS), replicates, blanks, and duplicates. This section provides an overview of the performance for each of these QA/QC sample types in terms of monitoring the nickel, cobalt, and iron fusion/XRF analyses, as well as whole rock (or "total") nickel by fusion/ICP.

11.4.1 Certified Reference Materials

Nickel analyses by fusion/XRF were mostly monitored by OREAS 72B, 74B, and 75B (Table 11-3). There are no failures for any of these CRM, although analyses do show a weak positive bias of +0.6 to 0.8 standard deviations (Figure 11-1). This was pointed out in previous technical reports (Voordouw and Simpson, 2018; Grandillo, et al., 2020); however, most of this bias was incurred from assays done between 2010 and 2012 with the most recent two programs (2021, 2022) showing no bias.

Results were similar for nickel by fusion/ICP, with 204 CRM returning no QA/QC failures and a weak positive bias of +0.4 standard deviations (Figure 11-1).

Cobalt analyses by fusion/XRF show 40 QA/QC failures out of 222 CRM for a very high failure rate of 18%. However, this failure rate is related to the low ratio of certified standard deviation (SD) to detection limit (DL) values, which is <1 in this case whereas ideally it is ≥10. This low ratio compromises the relevance of CRM for monitoring assays. Other summary statistics for CRM cobalt by fusion/XRF indicate reliable analyses, such as the strong similarity between measured and expected means as well as an average bias of just +0.2 standard deviations (Figure 11-1).

Iron analyses by fusion/XRF were monitored with 500 CRM analyses of which only 4 failed QA/QC, for a very low failure rate of 1%. Summary statistics show strong similarity between measured and expected means and a bias of +0.6 standard deviations (Figure 11-1). These CRMs indicate accurate and precise analyses for iron although within an analytical range (7-17% Fe) that is significantly lower than the associated magnetic separate samples (~30-50% Fe).

Table 11-3: Overview of CRM Performance (N >25) for the Decar Property

Element and Method	CRM	N	Meas.	Exp.	Exp. 1SD	Bias (SD)	Failures	Failure rate	SD/DL
ppm Ni by fusion/XRF dl = 30 PPM	OREAS 72B	96	7,229	7,086	178	0.8	0	0%	6
	OREAS 74B	67	34,452	33,933	932	0.6	0	0%	31
	OREAS 75B	65	54,415	53,800	1020	0.6	0	0%	34
ppm Ni by fusion/ICP dl = 100 ppm	OREAS 72B	83	7,141	7,050	253	0.4	0	0%	3
	OREAS 74B	67	34,415	34,286	1186	0.1	0	0%	12
	OREAS 75B	65	54,498	53,600	1800	0.5	0	0%	18
ppm Co by fusion/XRF dl = 40 ppm	OREAS 72B	94	136	126	24	0.4	0	0%	1
	OREAS 74B	66	509	502	12	0.6	25	38%	0
	OREAS 75B	62	781	788	18	-0.4	15	24%	0
% Fe by fusion/XRF dl = 0.01%	OREAS 72B	95	6.9	6.8	0.1	1.0	4	5%	20
	OREAS 74B	85	12.4	12.4	0.2	0.2	4	5%	20
	OREAS 75B	67	17.2	17.2	0.2	0.3	0	0%	20
	OREAS 13B	65	8.5	8.4	0.1	0.6	0	0%	10

Figure 11-1: Shewhart control charts of Z-scores for CRM used to track (clockwise from top left) Nickel by fusion/XRF, whole rock Ni by Fusion/ICP, Iron by Fusion/XRF, and Cobalt by Fusion-XRF

Note: Red and orange lines Show Z-scores of ±3 and ±2, respectively.
Source: Equity Exploration, 2022.

11.4.2 Non-certified Standards

Non-certified standards (NCS) are CRMs certified for analytical methods that are different from those used by FPX.

Nickel analyses by fusion/XRF were monitored with 424 non-certified standards (NCS). Results show that measured and expected means for these NCS are very similar, with low bias and zero failures (Table 11-4).

Whole rock nickel analyses by fusion/ICP show a failure rate of 4%. However, the cause of this increased failure rate again appears to be the low ratio of standard deviation to the detection limit. Other summary statistics for nickel by fusion/ICP indicate reliable analyses, such as the strong similarity between measured and expected means as well as an average bias of +0.4 standard deviations (Table 11-4).

Table 11-4: Overview of NCS Performance (N > 25) for Decar Property Analyses

Element & Method	CRM	N	Meas.	Exp.	Exp. 1SD	Bias (SD)	Failures	Failure rate	SD/DL
ppm Ni by fusion/XRF dl = 30 PPM	OREAS 13B	284	2,354	2,247	155	0.7	0	0%	5
	OREAS 73A	82	14,427	14,400	600	0.0	0	0%	20
	OREAS 75A	58	51,362	52,500	2,150	-0.5	0	0%	72
ppm Ni by fusion/ICP dl = 100 ppm	OREAS 13B	284	2,385	2,247	155	0.9	10	4%	2
ppm Co by fusion/XRF dl = 40 ppm	OREAS 13B	274	81	75	8	0.7	25	9%	0
	OREAS 73A	74	302	302	17	0.0	2	3%	0
	OREAS 75A	58	892	894	15	-0.1	15	26%	0
% Fe by fusion/XRF dl = 0.01%	OREAS 73A	82	9.2	9.2	0.2	-0.2	0	0%	15
	OREAS 75A	58	18.7	19.1	0.8	-0.4	0	0%	80

Source: Equity Exploration, 2022.

Cobalt analyses for NCS by fusion/XRF show a high failure rate (9%), but also a ratio of SD:DL that is ≤1, meaning these NCS are inadequate for monitoring QAQC for these analyses. Other summary statistics indicate reliable fusion/XRF cobalt analyses, including measured means that are within 2% of expected means for all NCS except the lowest grade standard (OREAS 13B) and a lack of analytical bias.

Iron analyses for NCS by fusion/XRF shows zero failures in 140 analyses. Measured and expected means are all within 2% of each other and there is no bias.

11.4.3 Replicates

Replicates are standard materials that were generated by FPX from drill core (07PXB028, 08RMB214, 13TAR001) and a 50 kg bulk sample (DTR). Unlike the CRMs, NCS, and blanks, the replicates are run through the Davis Tube magnetic separator and thus assess the whole analytical process. Means for replicates were calculated by averaging all samples within the second and third quartiles whereas the 1SD value was calculated by assuming a relative standard deviation (RSD) of approximately 3% or, if the RSD <3%, the detection limit. For comparison, the RSDs for CRMs range from 1-10% and average 4%.

Starting in 2011, the accuracy of Davis Tube magnetic separation was monitored with the DTR replicate developed by Cliffs from 50 kg of pulverized, awaruite-mineralized, ultramafic rock taken from the Baptiste Deposit (Ronacher, et al., 2013). A new set of internally derived DTR nickel replicates was used for the 2017 and 2021 programs and was made from outcrop samples of awaruite-mineralized peridotite collected from the Property in 2007 (07PXB028), 2008 (08RMB214), and 2013 (13TAR001).

DTR nickel analyses by fusion/XRF were monitored with 346 replicates. Results show that measured and expected means for these replicates are very similar, with no significant bias, and the failure rate is reasonable at 4% (Table 11-5).

Table 11-5: Overview of Replicate Performance (N > 25) For Decar Property Analyses

Element & Method	CRM	N	Meas.	Exp.	Exp. 1SD	Bias (SD)	Failures	Failure rate	SD/DL
ppm Ni by fusion/XRF dl = 30 PPM	08RMB214	38	6,198	6,230	187	-0.2	8	21%	6
	13TAR001	56	15,088	15,089	453	0.0	1	2%	15
	DTR	252	20,154	20,096	603	0.1	6	2%	20
ppm Ni by fusion/ICP dl = 100 ppm	08RMB214	30	1,493	1,506	100	-0.1	8	27%	1
	13TAR001	51	1,347	1,356	100	-0.1	3	6%	1
	DTR	171	2,271	2,261	100	0.1	23	13%	1
ppm Co by fusion/XRF dl = 40 ppm	08RMB214	38	406	412	40	-0.2	1	3%	1
	13TAR001	56	630	626	40	0.1	0	0%	1
	DTR	242	699	696	40	0.1	19	8%	1
% Fe by fusion/XRF dl = 0.01%	08RMB214	38	41.9	42.4	1.3	-0.4	7	18%	127
	DTR	252	37.7	37.8	1.1	-0.1	10	4%	113

11.4.4 Blanks

Cross-contamination at the crushing and/or pulverization stages was monitored with samples of blank material. In 2011 and 2012, this material is referred to as "quartz" and "quartz sand" (here assumed to be the same material) whereas blank used for the 2017, 2021, and 2022 programs was comprised of unmineralized bedrock from the Sitlika assemblage. In 2017, the Sitlika blank was also used to monitor cross-contamination within the Davis Tube.

Prior to sampling the Sitlika bedrock for blank material, outcrops were tested with a KT-10 magnetic susceptibility meter to ensure that they were non-magnetic. At the lab, pebble- to cobble-sized pieces of bedrock were then crushed and pulverized like the associated core samples, and then run through the Davis Tube magnetic separator to yield magnetic separates weighing between 0.0-0.1 g from a 30 g parent sample. In comparison, 30 g of mineralized peridotite typically yields close to 2.0 g of magnetic separate, or approximately 10-500 times that within the Sitlika bedrock.

The quartz sand used in 2011 and 2012 only assessed contamination in the pulverization stage since the sand would not undergo crushing during sample preparation.

Nickel analyses by fusion/XRF were done for 281 blanks, two of which returned more than the 10x detection limit threshold of 300 ppm (Figure 11-2). This failure rate of 1% is considered low and indicates minimal cross-contamination during sample preparation and analysis.

Nickel analyses by fusion/ICP were completed for 400 blanks, two of which exceeded the 10x detection limit threshold of 1,000 ppm. The Ni, Co, and Fe values returned by these two failed blanks, however, suggest they are likely mislabelled CRMs.

Cobalt by fusion/XRF likewise returned no failures out of 250 blanks analyzed with all results below the 40 ppm detection limit.

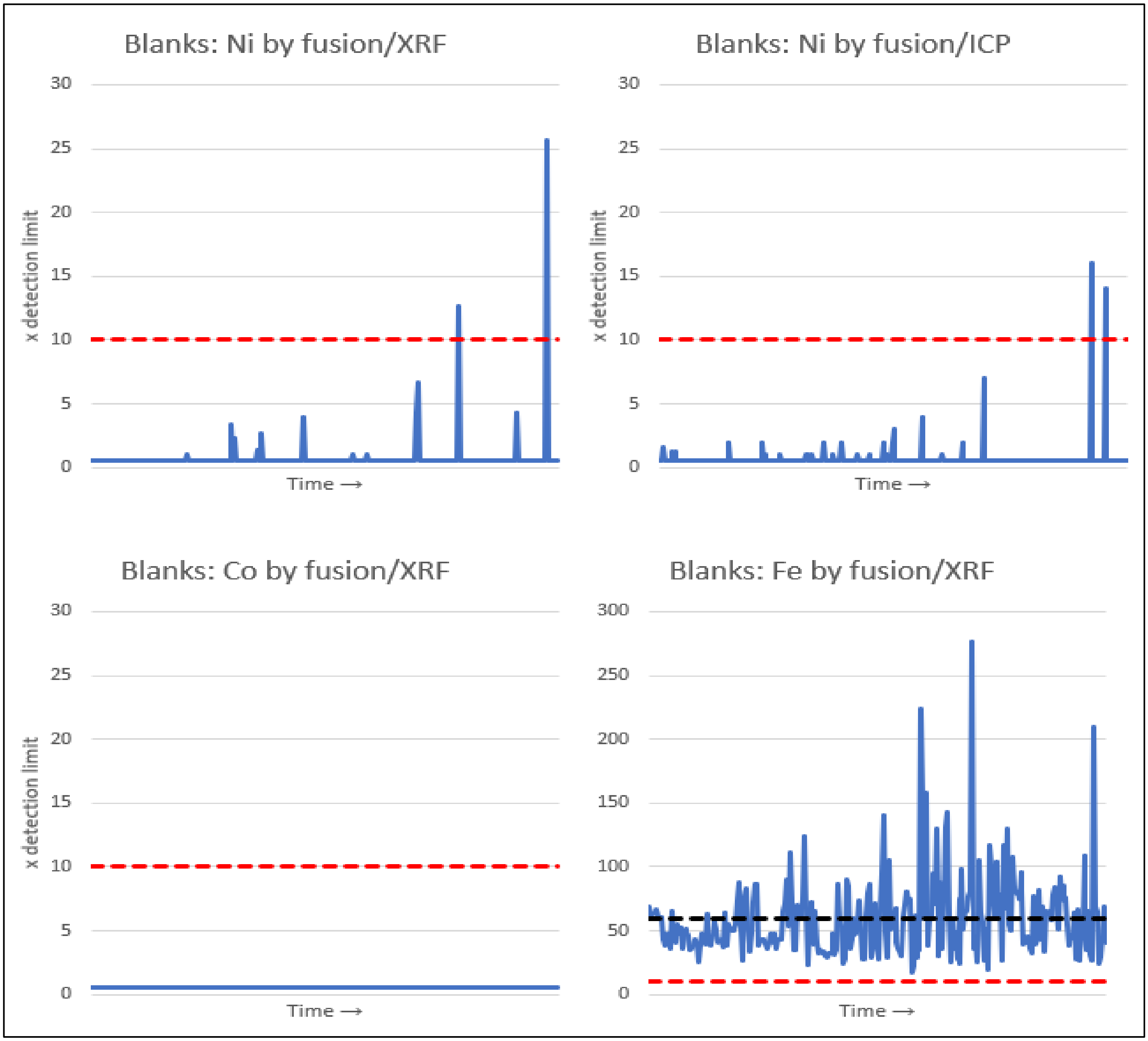
Iron analyses by fusion/XRF returned an average abundance of 0.59% Fe with a standard deviation of 0.31% Fe. This is significantly higher than the 0.01% Fe detection limit for fusion/XRF analyses so that the traditional metric for blank failures (i.e., 10x detection limit) cannot be applied in this case. An alternative method for assessing contamination is to compare the percent carryover, which commercial labs aim to keep ≤5%. In this case, the average quartz blank is assumed to contain 0.6% Fe and the average result for the magnetic separate core samples is 38% Fe. Percent carryover is then calculated as (100%*[a-0.6%]/38%), where “a” is any blank sample result, and shows that only one of the 281 blank samples exceeds the 5% carryover threshold.

The 16 blanks from the 2017 program that were run through the DTR stage did not return enough magnetic material to be analysed by fusion/XRF.

Table 11-6: Overview of Blank Performance for Select Decar Property Analyses

Element & Method	dl	N	Average	Failures	Failure rate
ppm Ni by fusion/XRF	30	281	<30	2	1%
ppm Ni by fusion/ICP	100	400	<100	2	1%
ppm Co by fusion/XRF	40	250	<40	0	0%
% Fe by fusion/XRF	0.01	281	0.59	1	0%

Figure 11-2: Control charts of detection limit multiples for blanks used to monitor contamination for (clockwise from top left) nickel by fusion/XRF, whole rock Ni by fusion/ICP, iron by fusion/XRF, and cobalt by Fusion-XRF.



Note: The red line shows the 10x detection limit threshold for typical QA/QC failure. The dashed black line on the Fe by fusion/XRF chart shows mean Fe.
Source: Equity Exploration, 2022.

11.4.5 Duplicates

Duplicate samples are used to quantify the reproducibility of core assay material and, for the Decar property, includes field (or quarter core), coarse (or preparation), pulp (done by the commercial lab), and check pulp (done at a different analytical lab). In general, there will be an increase in reproducibility through field, coarse, and pulp duplicates as the sample grain size is reduced and homogenized, quantified here by improved correlation (Figure 11-3) and the average coefficient of variance (CV_{avg}).

Field duplicates consist of quarter core collected on site that is then analyzed and compared against the parent half core assay. Summary statistics comparing parent and daughter assays for nickel by fusion/ICP and fusion/XRF, cobalt by fusion/XRF, and iron by fusion/XRF show generally strong correlation (Table 11-7). The CV_{avg} for field duplicates ranges from 9-12% and is analogous to other types of bulk tonnage deposits (Abzalov, 2008).

Coarse duplicates are produced at the lab by splitting off the parent between the coarse crushing and pulverization stages. Summary statistics of Decar coarse duplicates shows strong correlation for all analytes under review. The CV_{avg} ranges from 6-8% and as such is comparable to other types of bulk tonnage deposits (Abzaolov, 2008).

Pulp duplicates are produced at the lab by splitting off the parent after pulverization and show the same strong correlation for each analyte. CV_{avg} ranges from 5-6% which is, again, typical of bulk tonnage deposits (Abzalov, 2008).

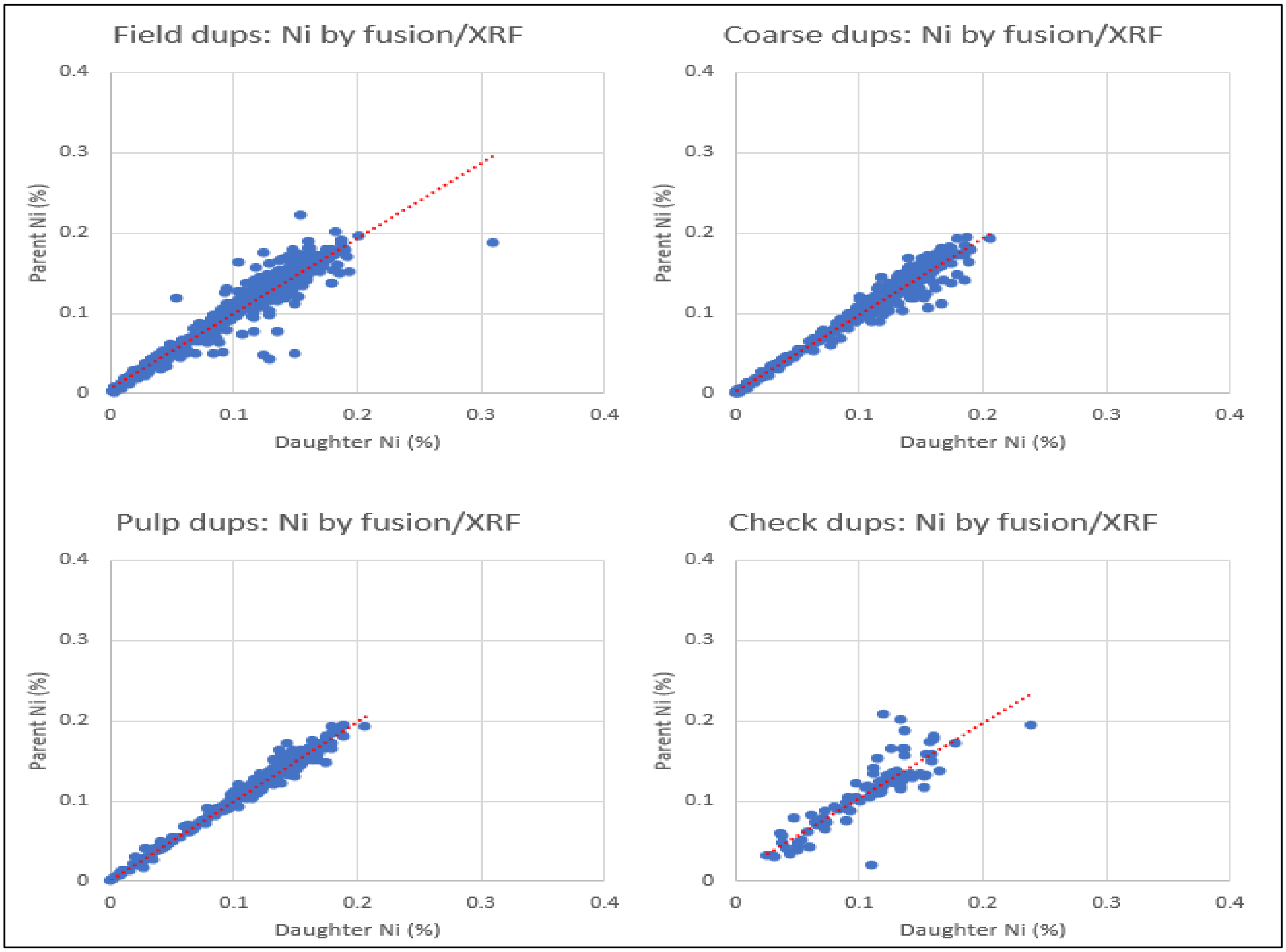
Check pulp duplicates were analyzed at SGS Canada in 2010, 2011, 2017, 2021, and 2022. The 2010, 2011, and 2021 check analyses include only nickel by fusion/ICP with results showing somewhat low correlation (0.61) and generally higher values in the SGS data, with an interquartile range of 2,010-2,300 ppm Ni compared to 1,890-2,200 ppm Ni for the parent. The somewhat poor correlation is at least partly related to the high detection limit (100 ppm) relative to the values being analyzed (~2,000 ppm), so that a difference of 1-2 detection units relates to a difference of 5-10% in grade. The 2017 and 2022 check assays were done on magnetic separates using fusion/XRF for Ni, Co, and Fe. Comparison of magnetic separate weights shows a much lower interquartile range for the 2017 check assays (8.6-13.6 grams in checks vs 11.1-15.3 g in parents) and significantly higher in the 2022 check assays (5.4-11.9 grams in checks, 4.6-8.3 g in parents). Review of the 2022 SGS methods showed that Davis Tube separation was done at 3,500 gauss (G) rather than the 3,300 G used by Actlabs; possibly the 2017 check assay program also used lower magnetic intensity for separation. As such, the SGS check assays are not true duplicates of the parent and so their Ni, Fe, and Co analyses are not directly comparable. This check assay work should be redone using the same Davis Tube separation settings for parent and duplicate samples.

Table 11-7: Summary Statistics for Duplicates Samples from the Decar Property

Element	Method	Unit	Type	Paris (N)	Parent Conc	Dup Conc	Correl	RSQ	CVavg
Ni	fusion/ICP	ppm	Field	604	1,985	1,987	0.93	0.87	9%
			Coarse	388	2,030	2,037	0.93	0.87	6%
			Pulp	403	2,023	2,025	0.94	0.89	6%
			Check	362	1,987	2,119	0.68	0.47	14%
Ni	fusion/XRF	ppm	Field	572	1,122	1,112	0.94	0.89	11%
			Coarse	366	1,128	1,101	0.98	0.95	8%
			Pulp	360	1,124	1,121	0.99	0.98	6%
			Check	85	N/A*	N/A*	N/A*	N/A*	N/A*
Co	fusion/XRF	ppm	Field	558	32	32	0.93	0.87	12%
			Coarse	352	33	33	0.97	0.94	8%
			Pulp	354	33	33	0.99	0.97	6%
			Check	65	N/A*	N/A*	N/A*	N/A*	N/A*
Fe	fusion/XRF	%	Field	558	2.3	2.3	0.95	0.89	10%
			Coarse	366	2.2	2.2	0.98	0.97	8%
			Pulp	360	2.2	2.2	1.00	0.99	5%
			Check	78	N/A*	N/A*	N/A*	N/A*	N/A*

* Data is not directly comparable as duplicate assays used different Davis Tube separation settings than the parent

Figure 11-3: X-Y Plots Showing Parent and Daughter Ni Assays for Field, Coarse, and Pulp Duplicates by Fusion-XRF as well as Magnetic Separate ("Mag Sep") Weights for Check Duplicates.



Note: The improvement in reproducibility from (top left) field duplicates through (top right) coarse duplicates and then (bottom left) pulp duplicates. Check assay magnetic separate weights are significantly lighter than the parent in the 2017 work and heavier in 2022 so that the fusion-XRF check assay results are not directly comparable. Red line shows correlation between parent and daughter pairs whereas the black line on the check dups plot tracks equal (i.e., 1:1) weights of parent and daughter separates.
Source: Equity Exploration, 2022.

11.4.6 Magnetic separates

A total of 1,494 magnetic separate duplicates were generated from field, coarse, pulp, and check duplicate samples. For those duplicates generated in the same lab, average weights were similar and correlations generally strong (Table 11-8). Check duplicates of magnetic separates, however, were on average 23% lighter in 2017 and 38% heavier in 2022, most likely due to the check assay lab using different Davis Tube separation settings than Actlabs.

Table 11-8: Summary Statistics for Magnetic Separate Duplicates from the Decar Property

Element	Method	Unit	Type	Pairs (N)	Average Parent (g)	Average Duplicate (g)	Correl	RSQ	CVavg
Mag Sus	DTR	g	Field	620	5.6	5.5	0.93	0.87	20%
			Coarse	397	5.6	5.4	0.99	0.98	14%
			Pulp	391	5.4	5.4	1.00	1.00	14%
			2017 check	21	13.5	10.4	0.69	0.48	29%
			2022 check	65	6.4	8.7	0.95	0.90	24%

11.4.7 Specific gravity

Specific gravity field duplicates were generated for 18 samples, with results showing strong correlation (R^2 = 0.9) with no indication of bias. Parent samples ranged from 2.6-3.0 g/cm^3, whereas duplicates show a range of 2.5-3.1 g/cm^3.

11.5 Data Adequacy

The core cutting, bagging, and transport procedures for all programs are industry standard. The authors are unaware of any security concerns related to drill core from the 2010, 2011, 2012, 2017, 2021, or 2022 programs.

Core sampling is comprehensive and consistent, with assays for 95% of all bedrock drilled and all analyses for Baptiste, Van, and the target B target conducted at the same commercial lab (Actlabs) using the same preparation and analytical methods. The only exception is core from the Sid target, which was analysed by ACME. Nickel analyses by fusion/XRF were monitored with a total of 2,124 QA/QC samples (CRM, blanks, field, and coarse duplicates) for an insertion rate of 15%, which falls within the recommended best practice range of 15-20% (Abzalov, 2008; Mendez, 2011). Adding in check assays and pulp duplicates analyzed by Actlabs increases the insertion rate to 18%.

It is the QP's opinion that the methods used to split, sample, secure and transport drill core are industry standard, and that the QA/QC procedures are in line with industry best practice. QA/QC sample insertion rates exceed industry best practice. The analytical methods and QA/QC results are also considered adequate.

Therefore, the geochemical assay data for the Decar property is considered adequate for the purposes of this Technical Report; however, recommendations for future geochemical assay program include the following:

1. Use CRM with Fe contents that more closely approximate the average Fe content of magnetic mineral separates (~38%).

2. Determine the reason for the positive bias in nickel analyses, possibly through a re-analysis of CRMs with methods used to certify them and those methods used to analyse the Decar sample.

12 DATA VERIFICATION

12.1 Legacy Data

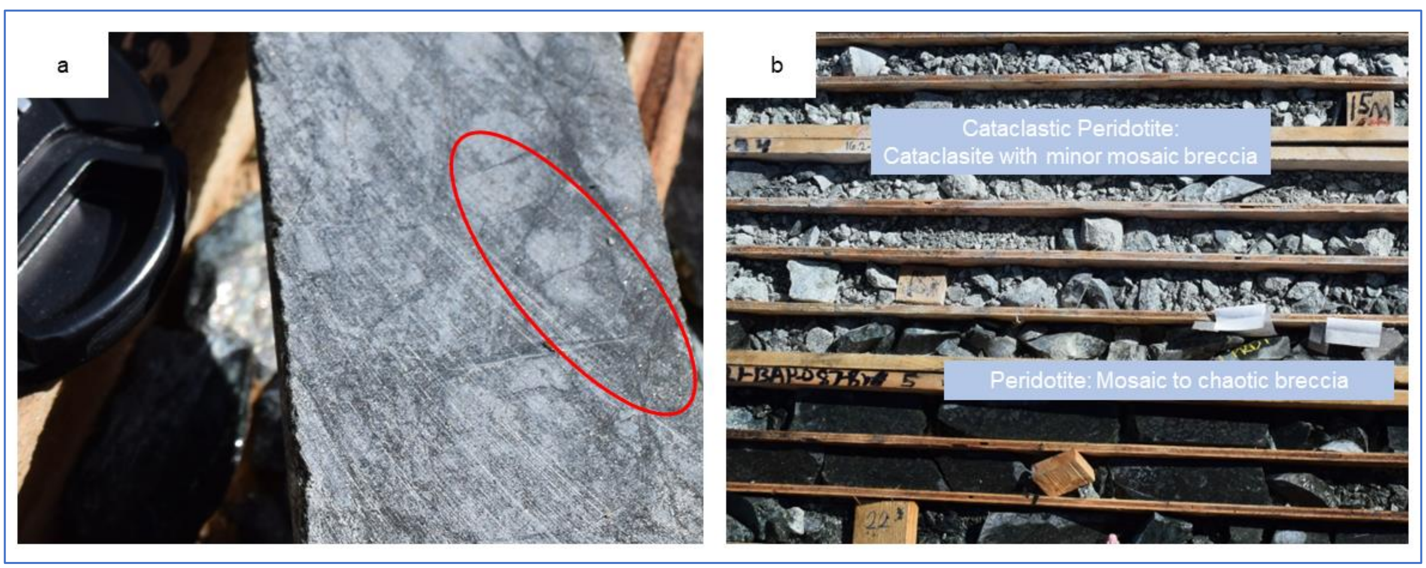
Data verification done by author Flynn included:

- a visit to the core yard at Fort St. James and the Decar Property, including the Baptiste Deposit, from July 4 to 7, 2022;
- 50 spot checks of assays in the database against lab certificates;
- field duplicate assays of 10 assay intervals; and
- review of QA/QC data compiled by author Voordouw in Section 11.

The first day of the 2022 site visit occurred two days prior to kick-off of the 2022 drill program. Authors Voordouw and Flynn liaised with Erin Wilson, Principal Geologist, FPX, who was on-site preparing for the 2022 drilling season.

July 3 was spent reviewing four 2021 bore holes: 21BAP084, 21BAP073, 21BAP085 and 21BAP087. Core was reviewed at the storage facility in Fort St. James and all holes were reviewed to observe zones of varying DTR nickel grades ranging from 0.04% to 0.23%. Core was noted for visible awaruite (Figure 12-1a) as well as varying zones of alteration and or deformation as per mineralized peridotite, cataclastic peridotite, and hornfels peridotite (Figure 12-1b).

Figure 12-1: Images of Core from Hole BAP21087

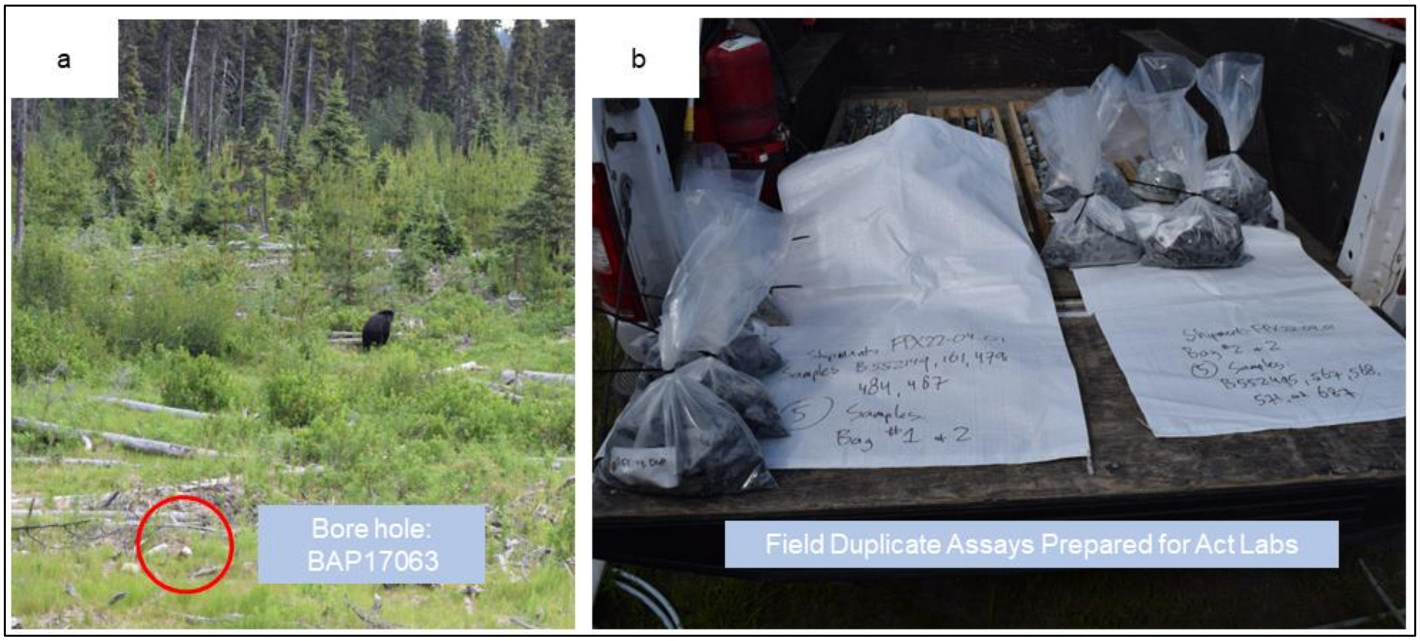


Note: Hole BAP21087 showing: a) visible grains of awaruite within mineralized peridotite, and b) cataclastic peridotite from 3 to 19.1m and mineralized peridotite below.
Source: Next Mine, 2022.

The Baptiste Deposit was traversed on July 4, 2022, to confirm the location of 2021 and historical drill collars (Figure 12-2a). Collar locations were measured with a handheld Garmin GPSMAP62. Offsets between 8 collar locations measured by Voordouw and Flynn and their surveyed locations in the database range from 0.3 – 7.0 m and average 3 m, which is within the ±5-10 m error typical for handheld GPS.

Ten intervals were selected as duplicate assays to be sent to Act Labs in Ancaster, ON, for analysis and comparison to the original assays for precision in the estimated variables of this report. Figure 12-2b illustrates the assays taken and Table 12-1, below, describes the comparison of the database ('original') assays versus the field duplicates taken July 5, 2022.

Figure 12-2: a) Reclaimed Drill Pad with Casing Circled, and b) Assays Taken as Field Duplicates for Processing at Act Labs



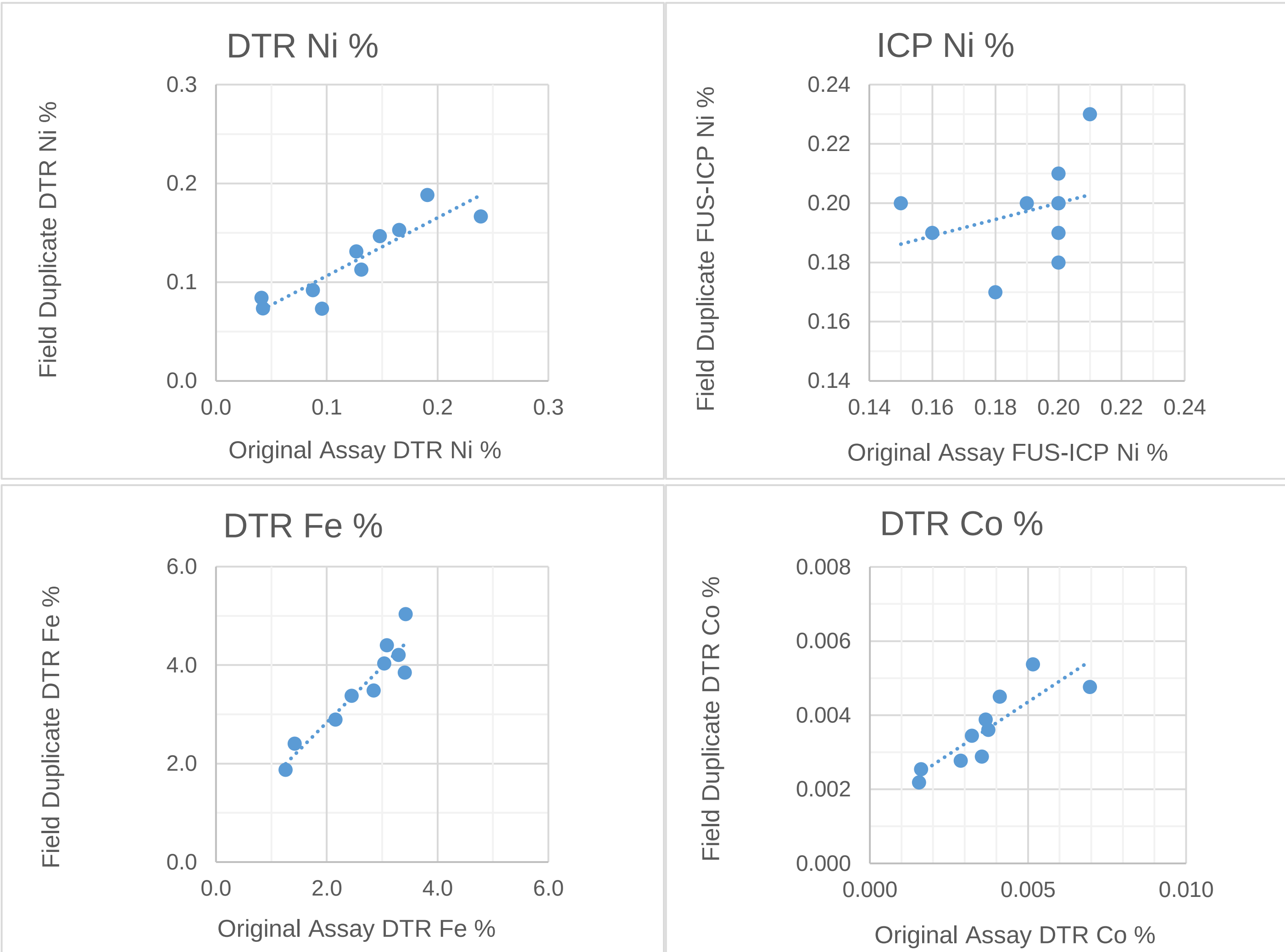
Source: Next Mine, 2022.

Table 12-1: Field Duplicate Assays Taken During Site Visit July 2022

Report Date: 3/8/2022					Original Assays				Field Duplicates		
Analyte Symbol				Ni	Ni	Fe	Co	Ni	Ni	Fe	Co
Unit Symbol				ppm	%	%	%	%	%	%	%
Detection Limit				0.01	0.01	0.01	0.004	0.01	0.01	0.01	0.004
Analysis Method				FS ICPES	DTR XRF	DTR XRF	DTR XRF	FS ICPES	DTR XRF	DTR XRF	DTR XRF
Sample	Hole ID	from	to								
B552149	21BAP073	250.0	254.0	0.20	0.19	3.42	0.005	0.21	0.19	5.03	0.005
B552161	21BAP073	285.0	287.0	0.18	0.17	3.29	0.004	0.17	0.15	4.21	0.004
B552479	21BAP084	58.0	62.0	0.21	0.15	3.08	0.004	0.23	0.15	4.40	0.005
B552484	21BAP084	73.0	75.6	0.19	0.13	2.16	0.003	0.20	-	-	-
B552484	21BAP084	73.0	75.6	0.19	0.13	2.16	0.003	0.20	0.13	2.90	0.003
B552487	21BAP084	80.2	81.3	0.20	0.10	2.85	0.004	0.20	0.07	3.48	0.003
B552495	21BAP084	98.9	103.0	0.20	0.09	3.04	0.003	0.20	0.09	4.03	0.003
B552567	21BAP085	66.0	70.0	0.16	0.04	1.26	0.002	0.19	0.07	1.87	0.002
B552568	21BAP085	70.0	74.0	0.15	0.04	1.42	0.002	0.20	0.08	2.40	0.003
B552571	21BAP085	77.9	82.0	0.20	0.13	2.45	0.004	0.19	0.11	3.38	0.004
B552687	21BAP087	11.0	15.0	0.20	0.24	3.41	0.007	0.18	0.17	3.85	0.005

Plotting of the original versus field duplicates is presented below in Figure 12-3. Performance of the field duplicates is acceptable and is meant to show precision of the original assay values to be used for geological modelling and resource estimation. While a sample size of ten assays is not statistically significant to warrant a lack of precision or a potential bias, they do serve as a validation of elements of interest present in the original assays.

Figure 12-3: Scatterplots of Original vs Field duplicates for DTR Ni%, ICP Ni%, DTR Fe% and DTR Co%

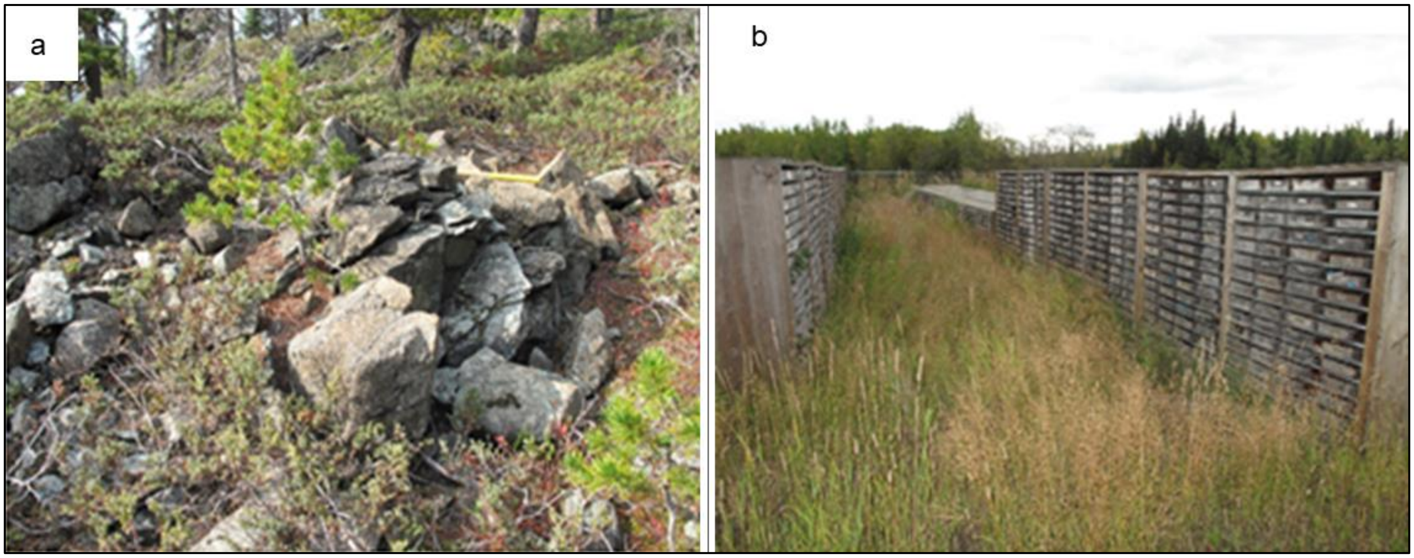


Source: Next Mine, 2022.

In the September 2018 site visit by Voordouw, several outcrops examined at the northwestern end of the Baptiste Deposit were found to consist of peridotite and dunite (Figure 12-4a). Other outcrops found along the road include argillite and mafic volcanic of the Sitlika assemblage, the distribution of which is consistent with property-scale mapping.

The historical core yard in Fort St. James was visited (Figure 12-4b), with the aim of cataloguing the inventory and reviewing some key intervals of drill core. The inventory was verified at approximately 30,200 m of core, which is consistent with the sum of metres drilled in 2010, 2011, 2012, and 2017. The historical core is clearly labeled and well preserved. Historical core intervals that were reviewed include 18-64 m from 11BAP008, 22-39 m from 11BAP027, and 45-125 m from 12BAP043. The interval in 11BAP008 included the Sitlika argillite and mafic volcanic that bound the Baptiste Deposit to the southwest, 11BAP027 included an internal of (low-grade) mineralized dunite, and 12BAP043 comprised one of the higher-grade DTR nickel intervals intersected at Baptiste.

Figure 12-4: Photos from Voordouw's 2018 Site Visit



Note: Photos show (a) outcrop of peridotite at northwest end of Baptiste Deposit, and (b) part of the historical core storage yard in Fort St. James.
Source: Equity Exploration, 2018.

12.2 Verification performed by the QP

The 2018 and 2022 site visits and review of FPX data and management of the 2017 and 2021 drill programs by Equity Exploration have provided the necessary level of confidence that data from the Decar Project is adequate for the purposes of this report.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

A significant volume of metallurgical testwork has been completed toward the development of a successful process design for Baptiste. This work has spanned approximately 13 years at numerous metallurgical test facilities using representative sample materials that have been sourced from the Baptiste Deposit. Table 13-1 outlines the various testwork programs, sample materials, and dates of testing for the metallurgical development.

The primary economic nickel mineral in the Baptiste Deposit is awaruite, which is a metallic nickel-iron alloy that is characterized by strong ferromagnetism (up to 10 times greater magnetic susceptibility than magnetite), high density (specific gravity of 8.6), and high nickel content (~76% Ni). As there are no existing industrial-scale operations that process awaruite-dominant nickel mineralization, a primary objective of the metallurgical testwork was to define a flowsheet using industry standard unit operations selected to take advantage of awaruite's unique properties.

Building on the metallurgical basis defined by historical testwork, the primary objectives of the prefeasibility level metallurgical program were to:

- Increase the understanding of the metallurgical response of Baptiste ore to the proposed processing methods by:
    - Conducting metallurgical tests on a range of samples from the Baptiste Deposit to verify variability of metallurgical response across the LOM;
    - Developing metallurgical recovery models to inform recovery predictions applied to the PFS economic model; and
    - Conducting pilot-scale grinding and magnetic separation testing to quantity the beneficial impact of preferential grinding of awaruite when grinding in closed circuit with a hydrocyclone.
- Refine the operating parameters of major unit operations to improve overall circuit performance.
- Provide input data for the process plant design and operating cost estimate.
- Generate bulk quantities of awaruite concentrate for detailed characterization and for additional downstream testing for refining awaruite concentrate into nickel sulphate suitable for use in the electric vehicle battery supply chain. Note that this section summarizes the mineral processing testwork supporting the Base Case of producing an awaruite concentrate and a nickel-rich MHP, while the metallurgical testwork for awaruite concentrate refining to nickel sulphate is summarized in Section 24.

Table 13-1: Metallurgical Testwork Summary Table

Year	Laboratory/Location	Major Testwork performed	Metallurgical Sample	Sample Size
2010	Knelson, Langley	Gravity separation	Individual drill intervals	20 kg
2011	SGS, Lakefield	Magnetic and gravity separation	2011 master composite	900 kg
2012	ALS Metallurgy, Kamloops	Magnetic and gravity separation pilot plant	Outcrop	23 t
2020	ALS Metallurgy, Kamloops	Magnetic separation / Flotation	PEA starter pit composite	500 kg
2021	ALS Metallurgy, Kamloops	Comminution characterization (SAB circuit)	Mine phase composites	7 x 250 kg
2021	Corem, Québec City	Comminution characterization (HPGR circuit)	Composite of mine phase composites	400 kg
2021	UBC, Vancouver	Comminution characterization (HPGR circuit)	Composite of mine phase composites	1.2 t
2021	SGS, Lakefield	Magnetic separation pilot plant	2021 LOM Composite	3.5 t
2022	ALS Metallurgy, Kamloops	Mineralogy	Mine phase composites	<10 kg
2022	SGS, Lakefield	Comminution characterization (SAB circuit)	Mine phase composites	3 x 100 kg
2022	SGS, Burnaby	Flotation / Flot. Tailings leach	Magnetic conc. from 2021 SGS Lakefield pilot plant	50 kg
2022	Corem, Québec City	Magnetic separation pilot plant	Outcrop	16 t
2023	Corem, Québec City	Magnetic separation variability testing	Mine phase composites / Outcrop / 2021 LOM Composite	8 x 20 kg

13.1.1 Metallurgical Development History

Historical metallurgical developments on the Baptiste Deposit are summarized in this section. Historical metallurgical development work detailed in earlier technical reports is not reported in detail in this report if they either a) pertain to processing methods which are no longer being pursued or b) have largely been superseded by more detailed or more representative work.

13.1.1.1 2013 Preliminary Economic Assessment by Cliffs

In the early 2010's, Cliffs Natural Resources (the former joint venture partner of FPX, then known as First Point Minerals Corp.) managed a testwork program to determine the major mineralogical characteristics of the samples and to provide preliminary separation and nickel upgrading testwork. Magnetic and gravity processes were tested extensively in attempts to upgrade nickel values to marketable concentrate grades. This was subsequently followed by a pilot plant program, conducted at ALS Metallurgy, aimed primarily at producing a sizeable concentrate sample to be used for product marketability development. This metallurgical development culminated in the 2013 Preliminary Economic Assessment which envisioned a 13.5% Ni concentrate at 82% DTR Ni recovery. The concentrate was identified as an intermediate product suitable as a feedstock into a ferronickel smelter or nickel sulphide smelter.

13.1.1.2 2020 Preliminary Economic Assessment by FPX

In early 2018, a review of previous testwork for the Baptiste Project was completed and a new suite of testwork was initiated at ALS Metallurgy. Testwork conducted was focused on magnetic separation to produce a magnetic concentrate, followed by froth flotation to recover an awaruite concentrate.

The use of froth flotation for final upgrading of the nickel concentrate proved to be able to produce significantly higher concentrate grades than the gravity work conducted in 2013. This permitted the nickel concentrate to be used directly as an alloying element in stainless steel production, eliminating the need for intermediate processing. Another key differentiator between processing routes was a finer regrind size ahead of cleaner magnetic separation to improve the liberation of awaruite and magnetite, producing a higher purity magnetic concentrate to feed into the final flotation upgrading stage.

This metallurgical development culminated in the 2020 Preliminary Economic Assessment which envisioned a 63% Ni concentrate at 84.7% DTR Ni recovery, with the concentrate being suitable as an alloying element in stainless steel production as a substitute for traditional ferronickel.

13.1.1.3 2023 Preliminary Feasibility Study

The metallurgical development of this study maintained the basic processing route of the 2020 PEA, namely sequential magnetic separation followed by final upgrading of the magnetic concentrate via flotation to produce a high-grade nickel concentrate suitable as an alloying element in stainless steel production. Significant additional metallurgical testing was undertaken for this PFS, commensurate with the increased level of testing maturity recommended at the PFS-level of study definition.

Part of the optimization work included adding a processing circuit to recover residual nickel contained in the flotation tailings via leaching and subsequent precipitation to produce a coproduct of a nickel-rich MHP, in addition to the high-grade nickel flotation concentrate which remains the dominant form of recovered nickel. Furthermore, engineering optimization resulted in a change of comminution technology changing from HPGR crushing, as detailed in the 2020 PEA, to SAG milling detailed in this study.

This metallurgical development culminated in this economic study which envisions the production of two nickel products, including:

- Life-of-mine 82.2% DTR Ni recovery to a flotation concentrate grading 60% Ni, with the concentrate being suitable as an alloying element in stainless steel production as a substitute for traditional ferronickel.
- An additional 6.5% DTR Ni recovery to a MHP product grading 45% Ni. MHP is an industrially established nickel intermediate suitable for refining to nickel metal or nickel sulphate.

13.1.2 Metallurgical Accounting

The Baptiste resource model and metallurgical recovery predictions have both been defined in terms of DTR Ni. Using DTR grades is commonly applied for magnetically recoverable mineral deposits, such as magnetite iron ore deposits. This section describes the DTR Ni methodology in order to provide context to the DTR Ni recoveries presented later in this section.

A Davis tube is laboratory scale, wet magnetic separator used to separate magnetically recoverable material from a mineral sample pulverized to 95% passing 200 mesh (75 µm). An assay is completed on both the feed material and the magnetic fraction from the Davis tube to determine the percentage of nickel which is magnetically recoverable. This percentage is then multiplied by the feed sample grade to determine the DTR Ni grade. The resultant DTR Ni grade should be considered as a small-scale metallurgical test rather than a traditional assay. For the Baptiste Deposit, the

DTR Ni measurement serves as a geometallurgical screening tool since a material proportion of total nickel exists in a form that is not recoverable by the proposed processing methods. This includes not only non-magnetic nickel minerals, but also very fine grained awaruite (<5 µm) which is not recovered in the DTR procedure. While the portion of DTR Ni varies throughout the Baptiste Deposit depending on the ratio of non-magnetic or very fine grained liberated awaruite, the DTR Ni metallurgical performance is generally consistent.

DTR Ni measurements were conducted for all magnetic separation products generated during PFS metallurgical testing. This permitted metallurgical accounting to be conducted on the DTR Ni basis to determine DTR Ni recovery. Metallurgical accounting was also conducted on a total nickel basis, with total nickel as determined by traditional assay and metallurgical accounting methods. This additional set of metallurgical accounting on total nickel basis served as a check on the validity of the DTR Ni metallurgical accounting methodology. To ensure equivalency of DTR Ni measurements between geological resource samples and metallurgical samples, the metallurgical test laboratories used the same DTR procedures applied to geological samples. The DTR Ni metallurgical accounting methodology was only applied to magnetic separation unit operations. For flotation and flotation tailings leach metallurgical testing, metallurgical accounting was only conducted using total nickel and the resultant total nickel recovery was assumed to be equivalent to the DTR Ni recovery. This is a reasonable assumption as the DTR Ni grade must be less than or equal to the total nickel grade.

Although the DTR procedure is a powerful geometallurgical tool, it is an imperfect proxy for metallurgical recovery since a) the magnetic separation conditions in the Davis tube differ from those uses in industrial magnetic separators and b) the Davis tube can capture non-magnetic nickel in gangue minerals not liberated from magnetic minerals and can capture nickel associated with magnetite which is purposely rejecting in the proposed Baptiste processing methods. Extensive bench and pilot-scale testing using unit operations that simulate industrial operation were used to define the relationship between DTR Ni and the metallurgical recovery projections of nickel.

Davis Tube Recoverable iron (DTR Fe) has been tracked in same manner as DTR Ni.

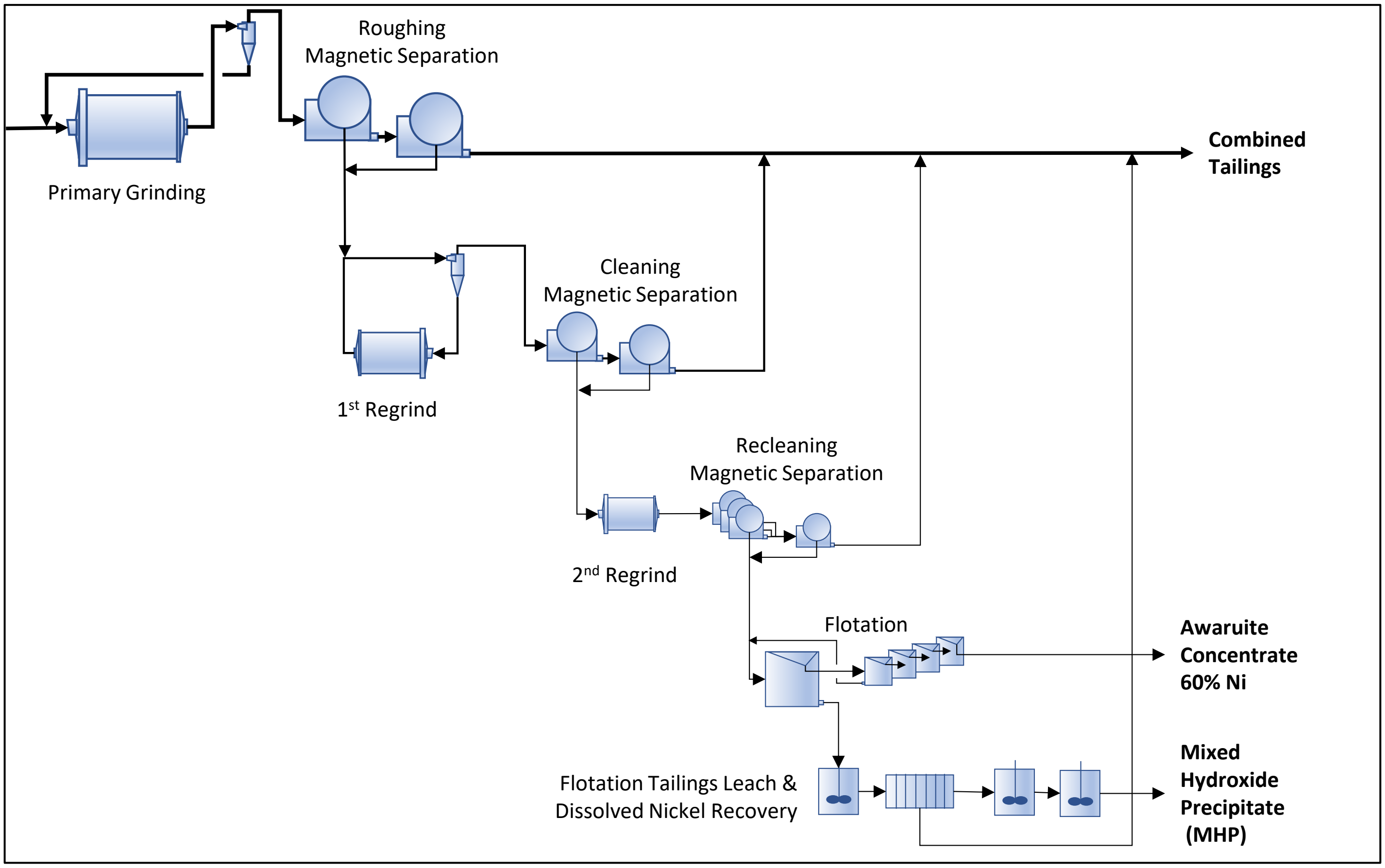
13.2 Summary of Metallurgical Test Results

The understanding of the metallurgical process requirements for the recovery of nickel from the Baptiste Deposit has evolved over the past 14 years of metallurgical testing, at various facilities using a number of representative samples. From the outset of testing, there has been a consistent reliance on magnetic processes for the recovery of awaruite and magnetite in a cost-effective rougher stage. The upgrading of nickel from the rougher stage requires regrinding of rougher concentrates to fine size distributions and includes several stages of magnetic upgrading to produce a magnetic concentrate that is predominantly magnetite and awaruite. This magnetic concentrate can be used as feed to a flotation process to separate and recover nickel to a high-grade nickel concentrate. Minor nickel losses in the flotation tailings can be partially recovered using a simple atmospheric leach and precipitation processes.

The process relies on several key mineralogical characteristics of the Baptiste Deposit. Firstly, the nickel mineral awaruite is easily recovered using magnetic separation and this allows for the recovery of awaruite even in poorly liberated particles. This fact allows for the use of relatively coarse primary grind (250 µm) in the preliminary stages of nickel recovery. Secondly, the grain sizes observed for awaruite are very fine, and consistently in the range of 5-10 µm and this dictates that fine-grinding (15-25 µm) is required to properly liberate awaruite and allow for the production of high-grade nickel concentrates using flotation.

The proposed flowsheet for the Baptiste project is shown in Figure 13-1 and includes the aforementioned unit operations.

Figure 13-1: Block Flow Diagram of the Baptiste PFS Processing Route



Source: FPX, 2023.

The results of extensive metallurgical testing have shown that nickel values can be consistently recovered using the process flowsheet above in Figure 13-1. A significant variability program was completed in 2023 and this demonstrated the consistent metallurgical response of the Baptiste materials to the proposed process flowsheet.

As a low-grade, high tonnage project, the Baptiste test work programs have relied on low-cost magnetic separation technology. The development of a successful flotation process to generate high-value magnetic nickel concentrates has added significant value to the project and places the Baptiste Deposit in a unique marketing position. Saleable concentrates of approximately 60 percent nickel are expected from the project when operating.

Testwork using a life-of-mine composite sample represents the nickel recovery expected from the Baptiste Deposit and is summarized in Table 13-2. This metallurgical result is consistent with other samples tested within the variability test work program. Nickel values are accounted for using both a DTR methodology and a traditional total nickel methodology and the differences reflects the magnetic responses of different nickel minerals within the Baptiste Deposit.

Table 13-2: Metallurgical Results for Life-of-Mine Composite Sample

	Overall Distribution (%)				Grades (%)		
	Mass	DTR Ni	Total Ni	DTR Fe	DTR Ni	Total Ni	DTR Fe
Mill Feed	100	100	100	100	0.13	0.21	2.39
Magnetic concentrate	4.3	94.1	58.2	94.1	2.83	2.83	52.0
Non-magnetic tailings	95.7	5.9	41.8	5.9	0.008	0.09	0.15
Flotation Concentrate	0.2	82.2	50.9	1.8	60.0	60.0	24.6
Flotation Tailings	4.1	8.92	5.5	92.2	0.28	0.28	53.2
Mixed Hydroxide Precipitate	0.02	6.5	4.0	0	45	45	0
Leach Tailings	4.1	4.5	2.8	92.2	0.14	0.14	53.8
Combined Tailings	99.8	11.3	45.1	98.2	0.015	0.09	2.35

Note: Abbreviated mass balance shown. Refer Table 13-16 for complete balance.

13.3 Sample Selection

Metallurgical samples used for the 2020 PEA and PFS metallurgical test campaigns are summarized in Table 13-12. For comparison purposes, the average grades and lithologies of the PFS mine plan are also listed.

All metallurgical samples and the PFS mine plan are dominated by the peridotite lithology. Including the peridotite subcategories of cataclastic peridotite and massive peridotite, all samples tested contain between 80-100% peridotite. Other notable lithologies include weakly mineralized dunite, non-mineralized gabbro dykes and argillite, and fault zones. DTR Ni grades range from 0.12% to 0.16%. DTR Fe grades range from 1.7% to 3.1%. Mineralogy of the metallurgical samples is discussed further in Section 13.4.

There are broadly three categories of metallurgical samples used: a life-of-mine composite, a near surface bulk outcrop sample, and various composites representing different phases of the mine life. These three categories are discussed in turn in the subsequent sections.

Table 13-3: List of Samples Used In Metallurgical Testwork Campaigns

Sample Name	Head Grade (%)			Lithology (%)	Sample Source			
	DTR Ni	Total Ni	DTR Fe		Form	Holes	Intervals	Weight (kg)
PEA starter pit composite.	0.16	0.23	2.8	94% peridotite, 6% cataclastic peridotit e	Crushed HQ ½ Core	4	52	480
2021 LOM comp. - Main sample	0.12	0.22	2.3	84% peridotite, 6% massive peridotite, 5% dunite, 4% fault zone	Crushed NQ/HQ ½ Core	80	493	2,300
Outcrop	0.12	0.22	2.5	<90% peridotite, >10% gabbro dike	Near surface, bulk sample	-	-	-
Comp. A - Mine phase 1AB	0.14	0.20	2.9	100% peridotite	HQ ½ / ¼ Core	4	12	110
Comp. B - Mine phase 1CD	0.14	0.24	2.7	98% peridotite, 2% fault zone	NQ/HQ ½ Core	9	363	2,000
Comp. C - Mine phase 1E	0.14	0.24	2.5	87% peridotite, 13% fault zone	HQ ½ / ¼ Core	11	102	640
Comp. D - Mine phase 2ABC (NE)	0.11	0.23	1.7	69% peridotite, 11% massive peridotite , 9% dunite, 5% argillite	HQ ½ / ¼ Core	8	54	525
Comp. E - Mine phase 2ABC (NW)	0.14	0.25	2.8	80% peridotite, 16% dunite, 4% massive perid.	HQ ½ / ¼ Core	9	68	480
Comp. F - Mine phase 3ABC	0.12	0.25	2.8	80% peridotite, 16% cataclastic perid., 3% fault	HQ ½ / ¼ Core	7	84	395
Comp. X - First 7 years of mining	0.15	0.25	3.1	93% peridotite, 6% massive peridotite	HQ ½ / ¼ Core	13	31	315
12HG04A	-	-	-	100% peridotite	Full HQ Core	1	6	135
Mine phase master composite	-	-	-	84% peridotite, 5% massive peridotite, 4% dunite, 5% fault zone	Assembled from mine phase composite A-F			1,750
PFS mine plan - Average	0.13	0.21	2.39	82 peridotite, 4% cataclastic peridotit te, 4% dunite, 3% massive peridotite	N/A	N/A	N/A	N/A
PFS mine plan – Yearly min/max	0.11-0.15	0.21-0.22	1.94-2.84	N/A	N/A	N/A	N/A	N/A

13.3.1 2021 Life-of-Mine Composite

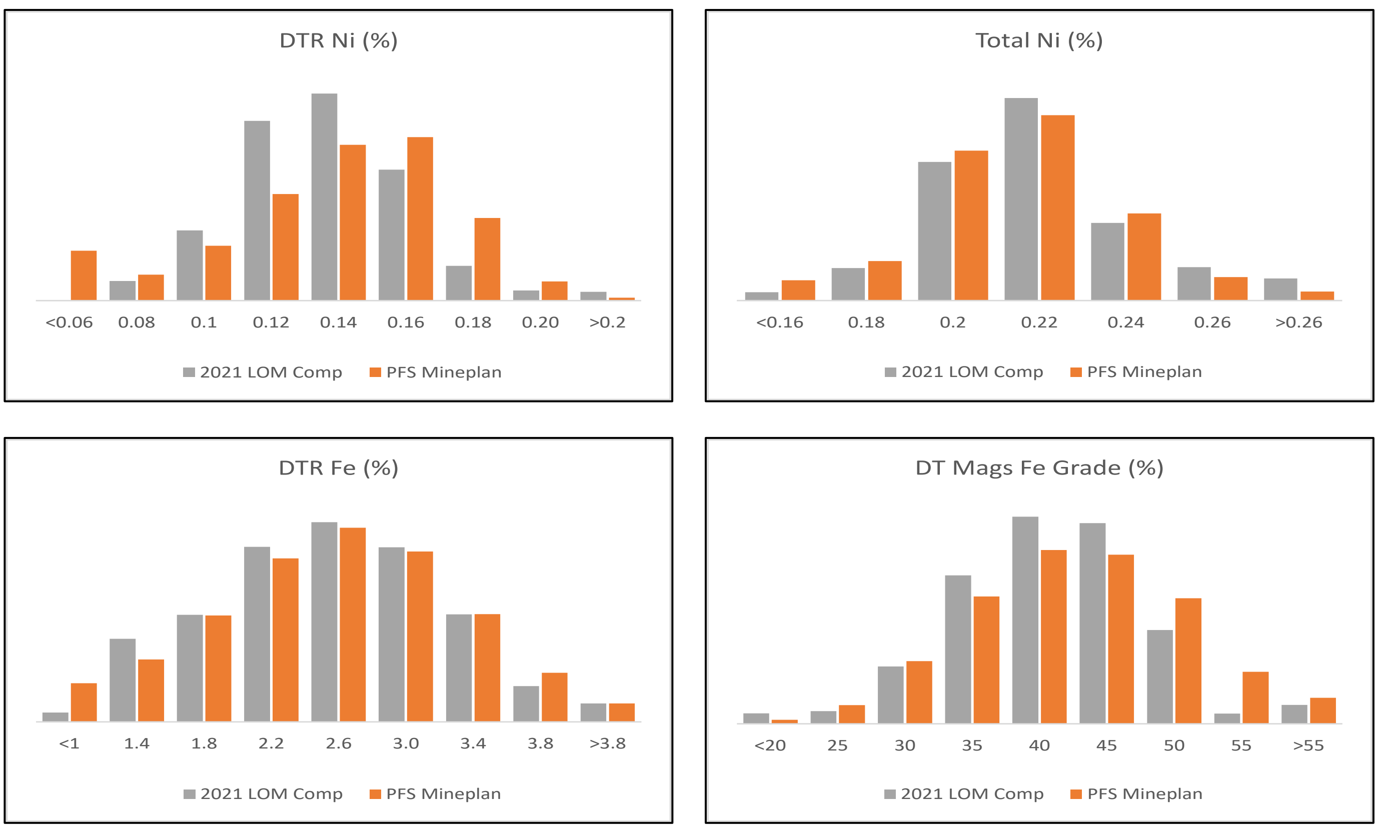
The 2021 LOM composite was assembled primarily for the 2021 magnetic separation pilot plant at SGS Lakefield, the subsequent bench-scale flotation program at SGS Burnaby, and various other small-scale programs. The composite was assembled from the crushed assay rejects of approximately 500 drill hole intervals to form a 2,300 kg composite. An additional 1,300 kg composite (not listed in Table 13-3) was assembled in a similar manner to commission the pilot plant prior to feeding the main sample.

The representativity of the resultant composite was assessed by comparing the distribution of various geometallurgical parameters of the life-of-mine composite to that of the PFS mine plan on an interval length weighted basis as follows:

- Figure 13-2 compares the DTR Ni, Total Ni, and DTR Fe grades, in addition to the iron grade from the Davis tube magnetic fraction, which is a proxy for the liberation and grain size of magnetite.
- Figure 13-3 compares rock and alteration parameters. Caution should be used when interpreting the alteration histogram since alteration logging differed between eras of geological drilling. An alternative metric for extent of serpentinization is loss on ignition (LOI) and is also presented in Figure 13-3.
- Figure 13-4 presents spatial parameters and the PFS mine phase. For reference, a diagram of the PFS mine phases is presented in Figure 13-6.

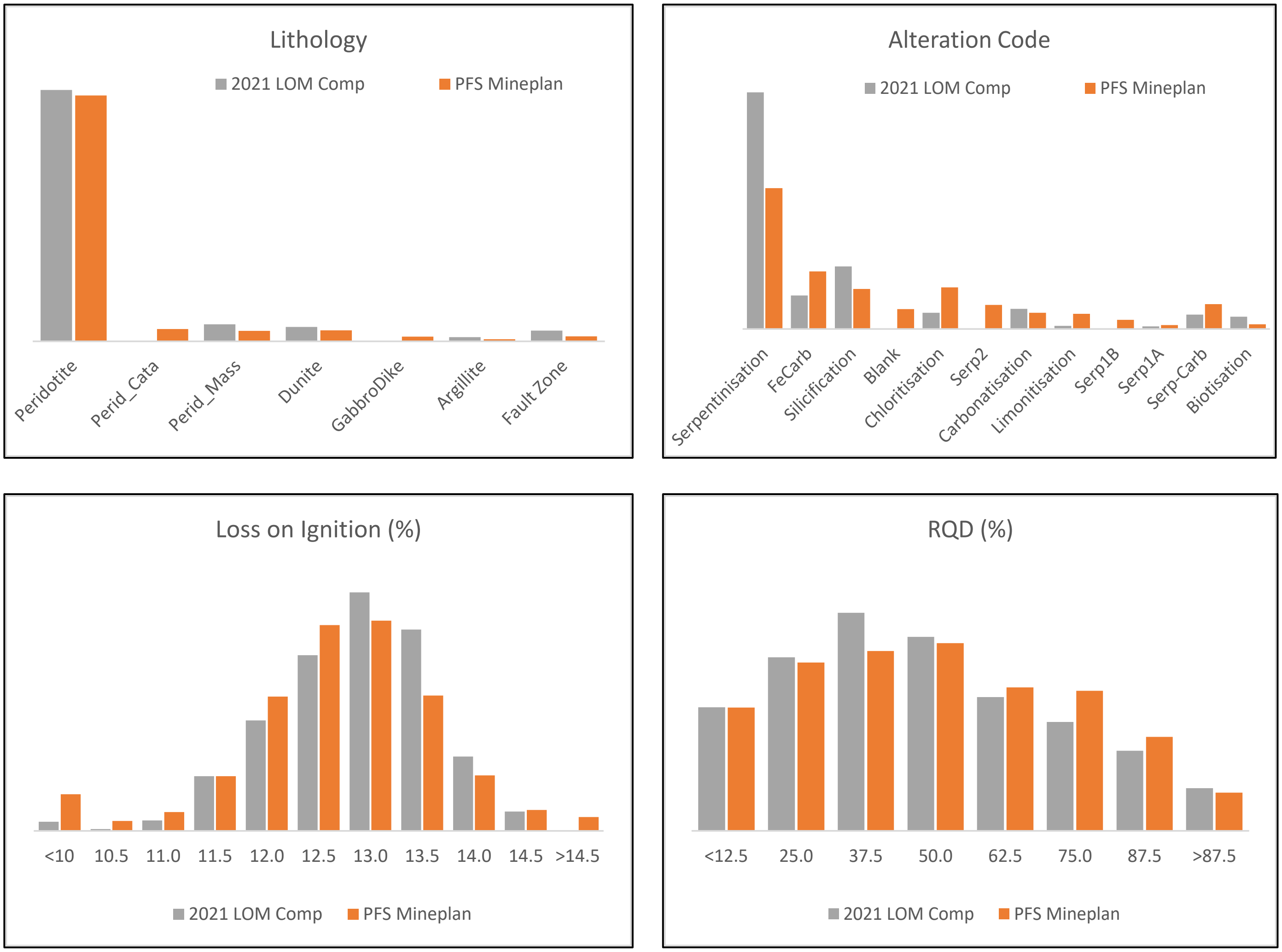
Based on this comparison of these characterization metrics it was assessed that the 2021 life-of-mine composite is reasonably representative of the PFS mine plan. This is reflective not just of the successful creation of the composite, but also of the relatively narrow range of distribution of geological properties throughout the Baptiste Deposit.

Figure 13-2: Histograms Comparing Grades of the 2021 LOM Composite to the PFS Mine Plan



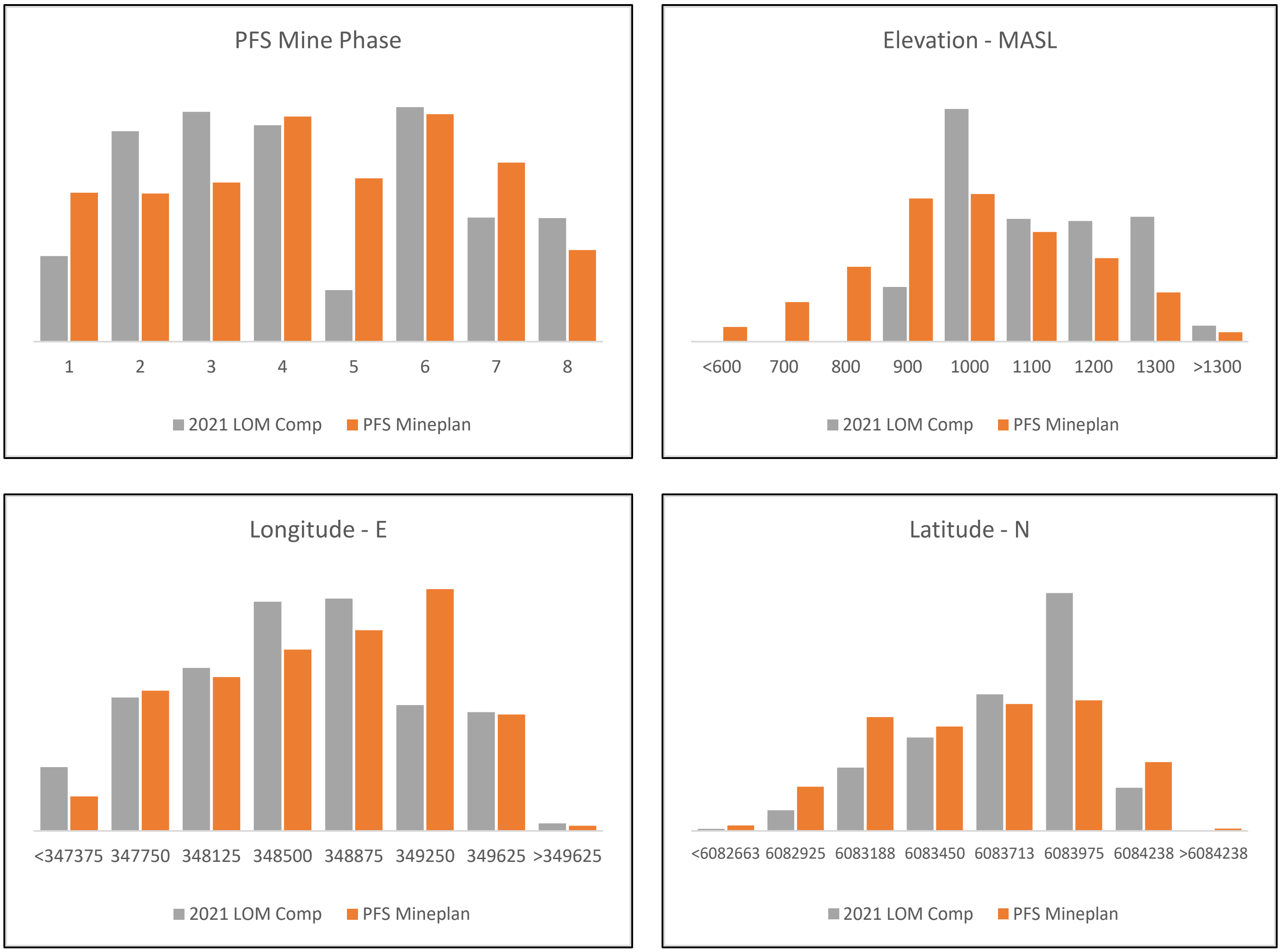
Source: FPX, 2023.

Figure 13-3: Histograms Comparing Rock and Alteration Properties of the 2021 LOM Composite to the PFS Mine Plan



Source: FPX, 2023.

Figure 13-4: Histograms Comparing Spatial Parameters of the 2021 LOM Composite to the PFS Mine Plan



Source: FPX, 2023.

13.3.2 Bulk Outcrop

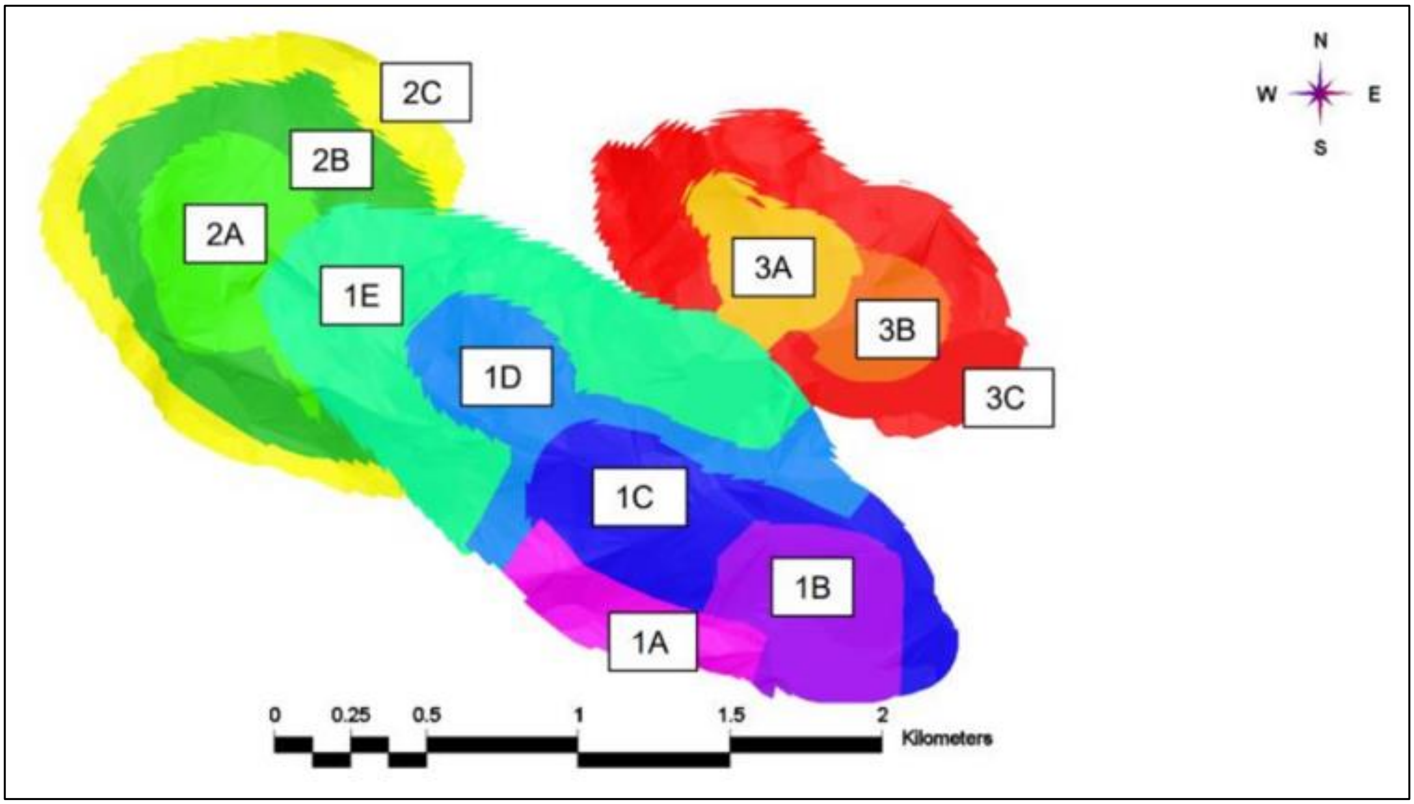
The bulk outcrop material was sourced from an excavation of near surface mineralization located in the northeast quadrant of Phase 1 of the PFS mine plan. A total of 250 t of mineralized peridotite was excavated in 2011 to provide feed to pilot-scale work completed at the time. The remainder of material was stockpiled at the fenced core storage facility in Fort St. James until sourced for the 2022 pilot plant at Corem. Reviewing of the lithology and grades (presented in Table 13-1) and mineralogy data (presented in Section 13.4) the bulk outcrop appears broadly comparable to other mine phases and the LOM material.

13.3.3 Mine Phase Composites

Selection of intervals for the mine phase composite were made based on the 2020 PEA mine plan, as the PFS metallurgical program commenced two years prior to development of the PFS mine plan. While the two mine plans are broadly similar, the nomenclature used for the mine phase composites metallurgical samples are reflective of the 2020 PEA nomenclature. Figure 13-5 presents the 2020 PEA mine plan and associated phases, while Figure 13-9 presents the PFS mine phases and associated phases. Figure 13-8 presents the distribution PFS mine plan material in mine phase composites and demonstrates that the composite sample still largely captures the different phases of mining. Table 13-4 presents the breakdown of plant feed by PFS mine phase.

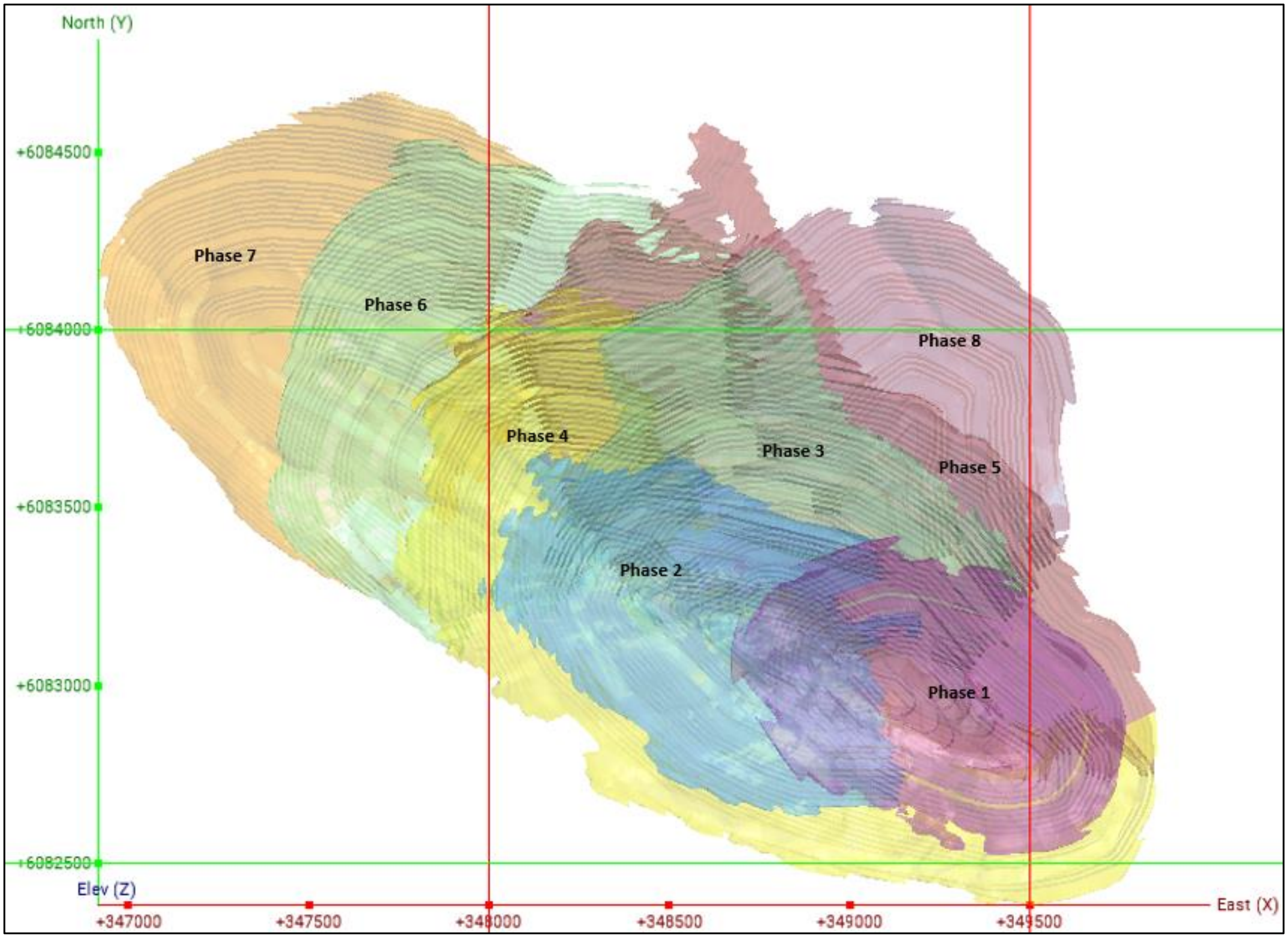
Reviewing various geometallurgical parameters against the PFS mine plan, no significant differences between the composites are noted, beyond those summarized in Table 13-3 and Section 13.4. The mine phase composites were used in the comminution characterization, mineralogy, and magnetic separation variability testing program.

Figure 13-5: 2020 PEA Mine Phase Pit Shells – Plan View



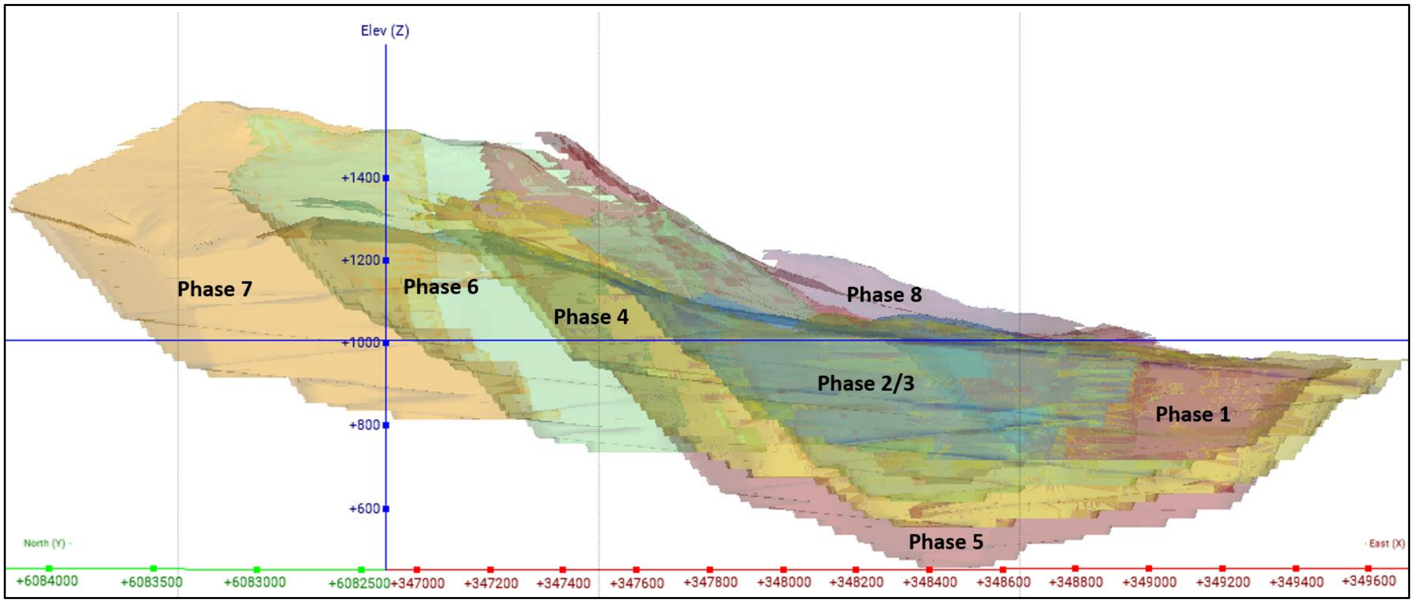
Source: Equity Exploration, 2020.

Figure 13-6: PFS Mine Phase Pit Shells – Plan View



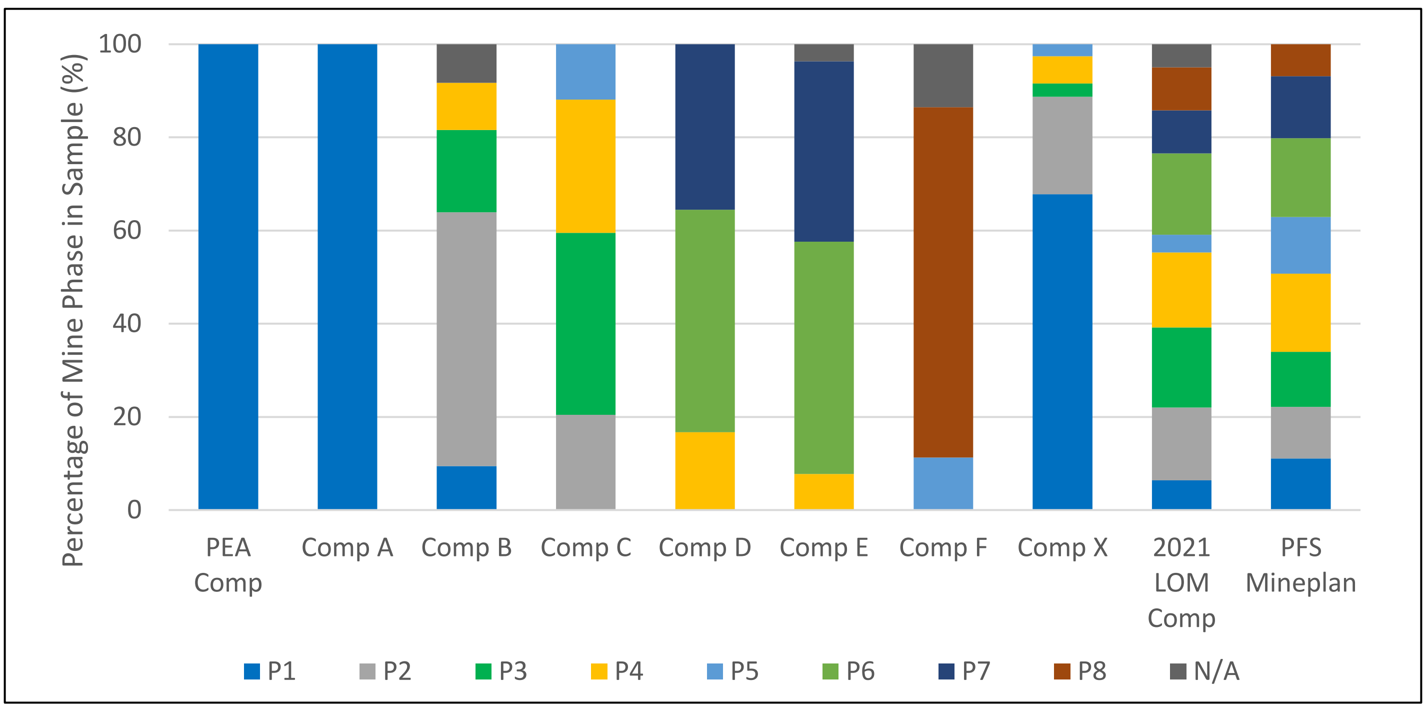
Source: FPX, 2023.

Figure 13-7: PFS Mine Phase Pit Shells – Looking North-East



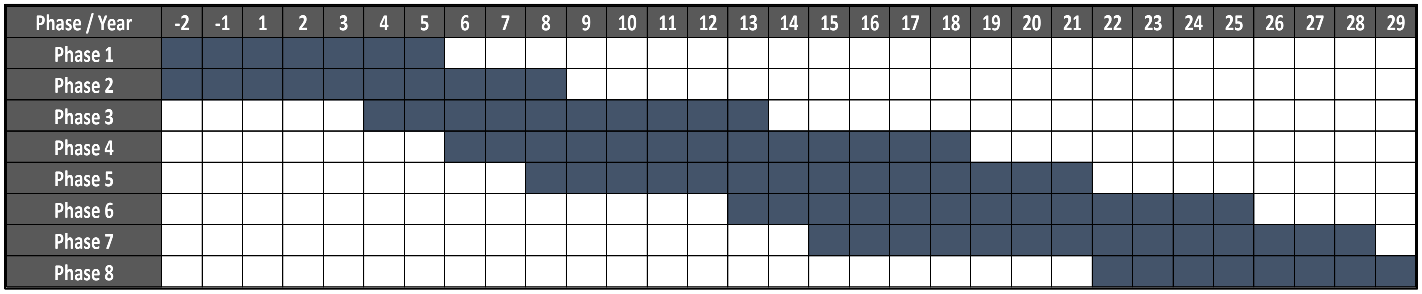
Source: FPX, 2023.

Figure 13-8: Distribution of Intervals Contained in the PFS Mine Phases in Mine Phase Composite Samples



Source: FPX, 2023.

Figure 13-9: Mine Phase Schedule



Source: TechSer, 2023.

13.4 Mineralogy

Mineralogy work was conducted at ALS Metallurgy on the mine phase composite, both pilot plant materials (2021 LOM composite and outcrop), and the PEA starter pit composite.

The objective of the mineralogy work was to:

- identify mineral species;
- determine awaruite and magnetite mineral grain size distributions and liberation characteristics/associations; and
- identify, quantify, and measure elemental compositions of nickel bearing minerals.

Particle mineral analysis was conducted by automated scanning electron microscopy (QEMSCAN). The QEMSCAN species identification protocol (SIP) was used to quantify mineral composition and an earlier metallurgical campaign confirmed this methodology was largely in line with mineral identification by x-ray diffraction. Elemental compositions were determined by energy-dispersive X-ray spectroscopy (Bruker X Flash).

Mineralogy completed on the PEA starter pit composite supersedes work done on the same material during the PEA phase of work, as the more recent work was run with a modified QEMSCAN SIP list which allowed identification of finer grains of awaruite. Identifying progressively finer awaruite grains, through extended scanning times leading to increased resolution, was a focus of the mineralogy work to differentiate between very fined grained awaruite and soluble nickel contained in silicate minerals. The mineralogy results from the PFS work program are broadly in line with those reported in earlier investigations.

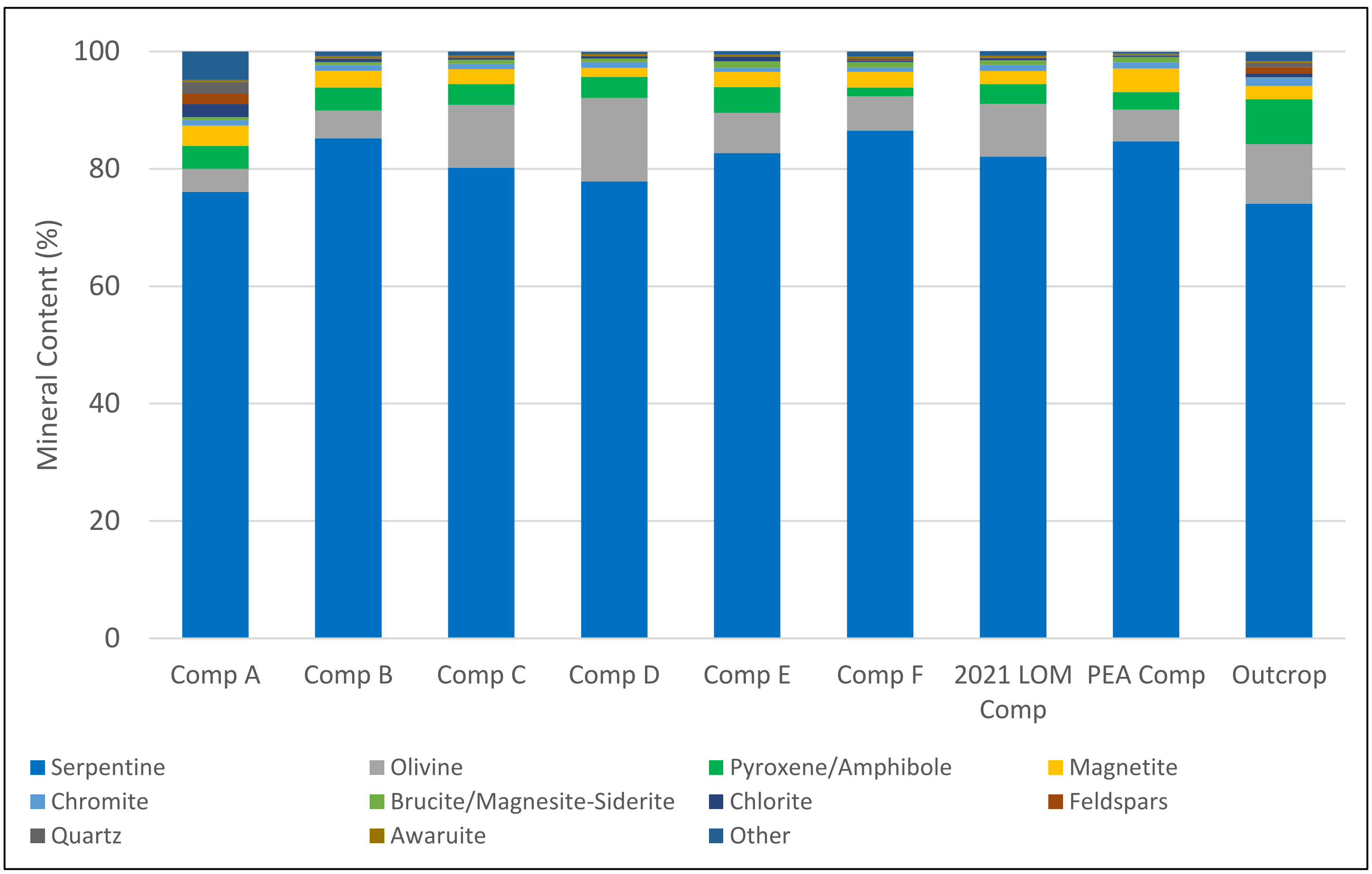
13.4.1 Mineral Content

Mineral content data is summarized in Figure 13-9 which confirms serpentine as the dominant mineral with minor amounts of olivine, pyroxene, magnetite, and magnesium/iron carbonates/hydroxides and trace amounts of awaruite and nickel sulphides. The QEMSCAN analysis does not distinguish between different polymorphs of serpentine. However, additional mineralogy, conducted at UBC using Raman spectroscopy, confirmed that lizardite is the predominant serpentine polymorph. Separate analysis for chrysotile, another serpentine polymorph and a known asbestiform mineral, indicates that the mine phase composites range from 0.50% to 1.75% chrysotile with a deposit average of approximately 0.9% chrysotile.

Nickel mineralization is summarized graphically in Figure 13-11 and tabulated in Table 13-4 along with Ni, DTR Ni, and sulphur grades. Nickel in all the composites was primarily contained in the form of awaruite, which contributed between 83% and 93% of the total nickel. The remainder of the nickel was present as sulphide minerals pentlandite and heazlewoodite. Overlaid onto Figure 13-11 is the percentage of total nickel reporting as DTR Ni. The awaruite content consistently exceeds the DTR Ni content indicating that a material proportion of awaruite is not captured by the DTR Ni procedure. This relationship is further discussed in Section 13.4.3 which reviews the impact of awaruite grain sizes on DTR Ni recovery.

The complete absence of any soluble nickel contained in silicate minerals contrasts with earlier investigations which determined a small percentage of total nickel reporting to silicates. This difference is attributed to more thorough mineralogical investigations in the PFS mineralogy program which differentiates between nickel in silicates and very fine grained awaruite. However, the conclusion of no nickel in silicates is suggestive rather than definitive. From a mineral processing perspective, this is immaterial as both nickel in silicates and very fine grained awaruite are not recoverable in the proposed processing methods. DTR Ni grades, mineralogy work, remain the best indicator of magnetically recoverable nickel.

Figure 13-10: Mineralogy Content of Metallurgical Samples

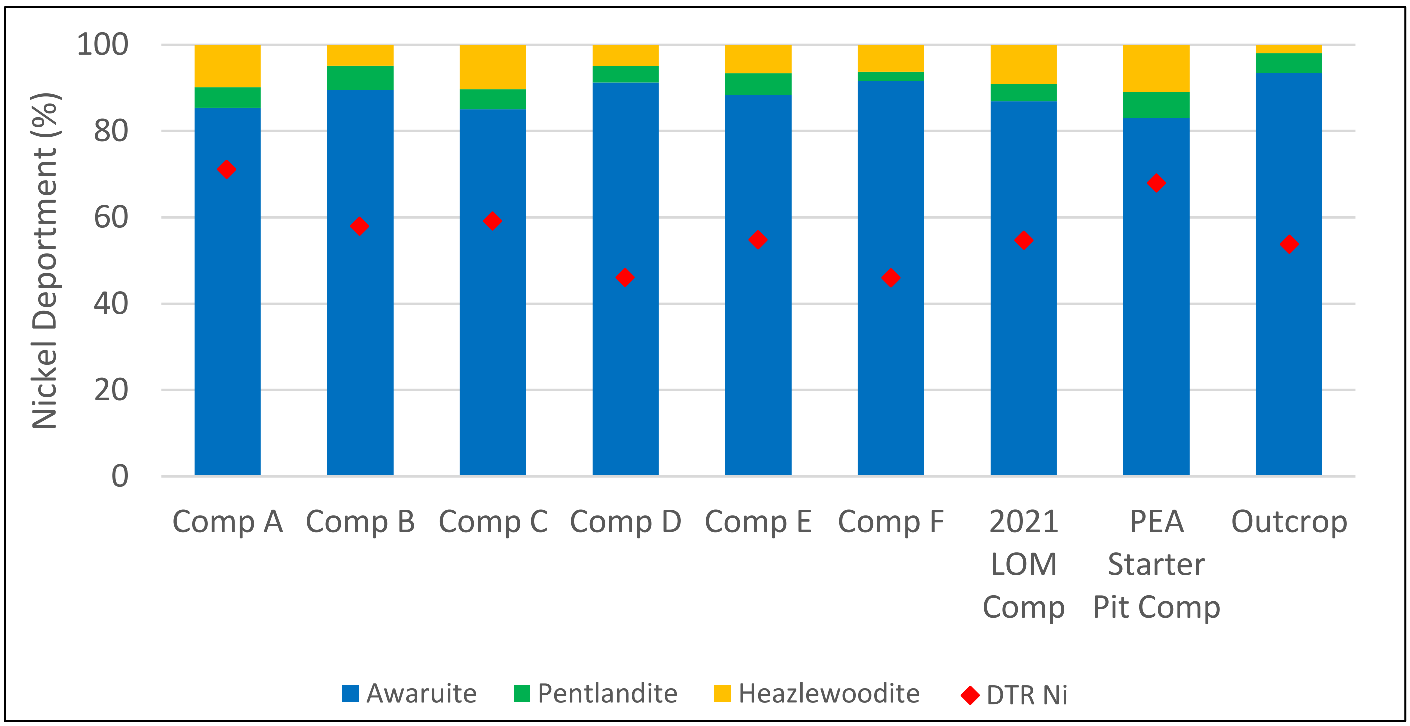


Source: FPX, 2023.

Table 13-4: Distribution of Nickel by Mineral in Metallurgical Samples

Mine Phase	Nickel Deportment (%)				Grade (%)		
	Awaruite	Pentlandite	Heazlewoodite	DTR Ni	Ni	DTR Ni	S
Comp. A - Mine Phase 1AB	85	5	10	71	0.20	0.14	0.10
Comp. B - Mine Phase 1CD	90	6	5	58	0.24	0.14	0.05
Comp. C - Mine Phase 1E	85	5	10	59	0.25	0.15	0.04
Comp. D - Mine Phase 2ABC (NE)	91	4	5	46	0.23	0.11	0.03
Comp. E - Mine Phase 2ABC (NW)	88	5	7	54	0.25	0.14	0.04
Comp. F - Mine Phase 3ABC	92	2	6	46	0.24	0.11	0.02
2021 LOM comp.	87	4	9	54	0.22	0.12	0.04
PEA Starter pit comp.	83	6	11	66	0.23	0.16	0.06
Outcrop	93	5	2	53	0.23	0.12	0.05

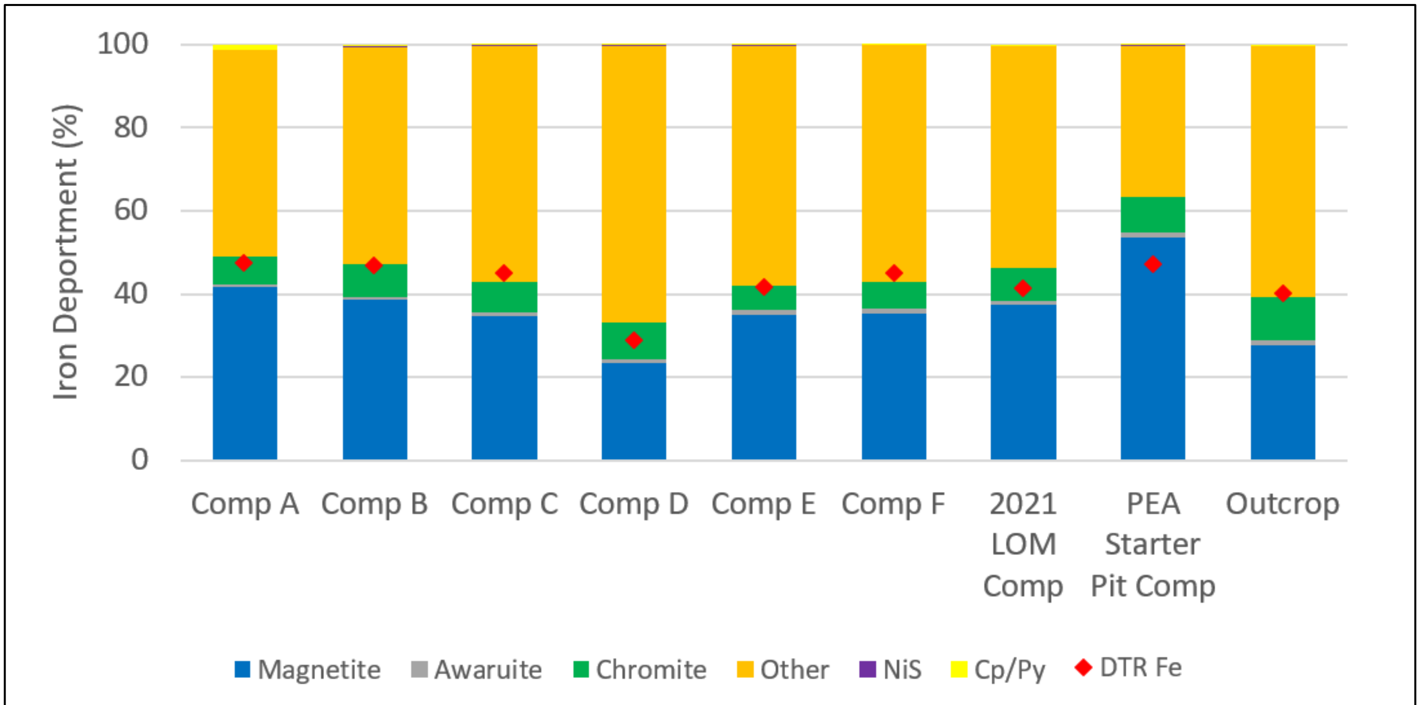
Figure 13-11: Distribution of Nickel by Mineral in Metallurgical Samples



Source: FPX, 2023.

In contrast to the relationship between DTR Ni and awaruite, DTR Fe did provide an indication of magnetite content in the composites. This is demonstrated in Figure 13-12 which shows a close correlation between DTR Fe content and iron reporting to magnetic minerals; namely magnetite, awaruite, and magnetic chromium containing species. Magnetite constituted approximately 1.5-4.0% of the composite masses; significantly higher than the awaruite contents which measured between 0.24-0.32%.

Figure 13-12: Distribution of Iron by Mineral in Metallurgical Samples



Source: FPX, 2023.

13.4.2 Elemental Analysis of Nickel Minerals

Spectra analysis of nickel containing minerals were completed by energy-dispersive X-ray spectroscopy. The results are presented in Table 13-5 and show an average nickel grade in awaruite of 76%. In addition to the results from the 2021 ALS Metallurgy program, results from a previous investigation conducted for Cliffs Natural Resources are included. This additional analysis shows the presence of cobalt and copper within the awaruite mineralization, which is suspected to be solid substitution of nickel or iron. The absence of these elements in the 2021 ALS mineralogy program reflects the methods utilized, rather than an absence of these elements which have consistently been reported in the final flotation concentrates (refer to Section 13.10).

Table 13-5: Average Elemental Grades of Nickel Minerals from the Mine Phase Composite Samples

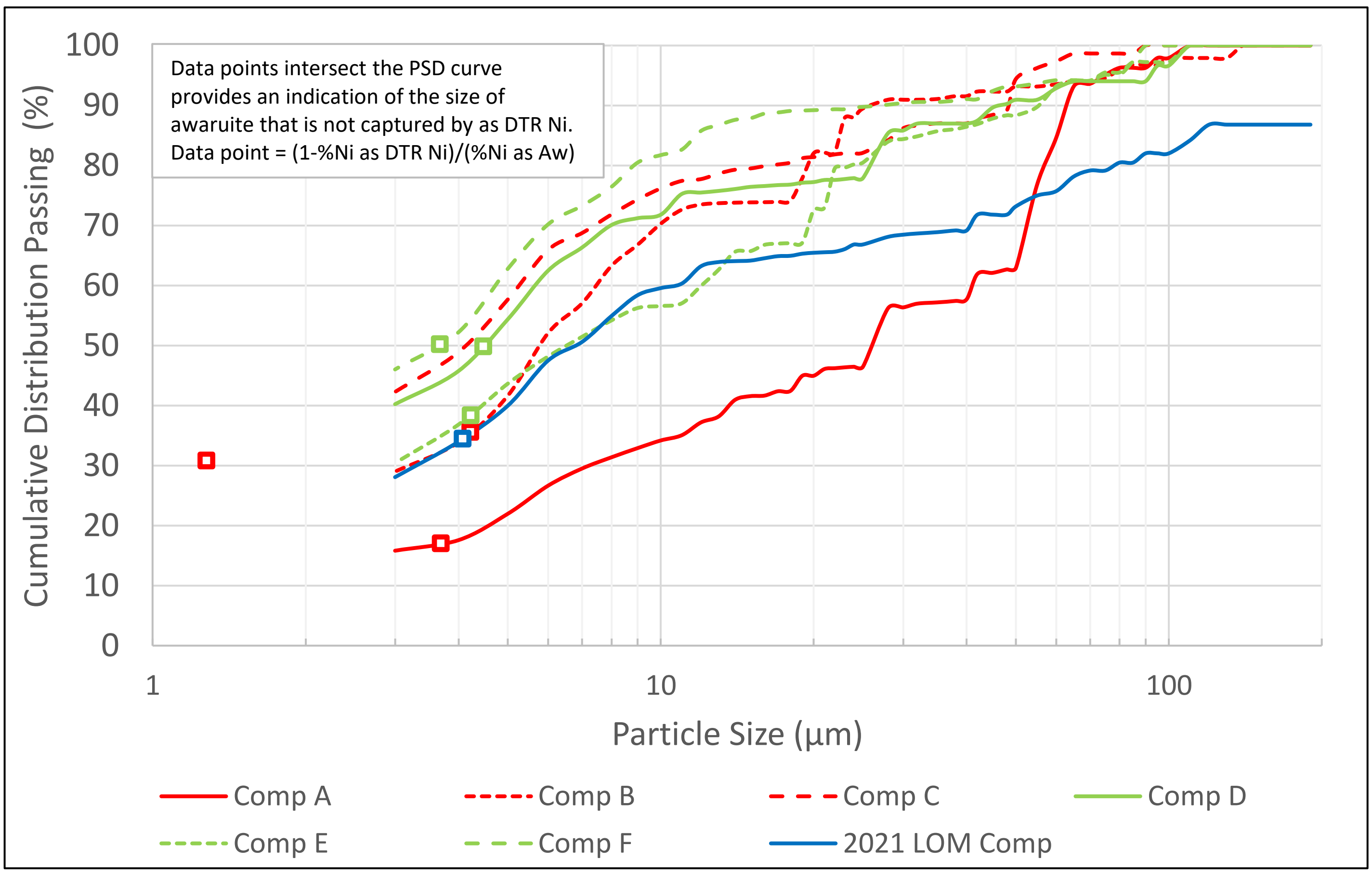
Element Grade (%)	Fe	Ni	S	Co	Cu
Awaruite (2021)	23.8	76.2	-	-	-
Awaruite (2011)	21.9	76.1	-	0.8	1.2
Heazlewoodite (2021)	1.6	73.2	25.2	-	-
Pentlandite (2021)	33.5	35.1	31.4	-	-

13.4.3 Mineral Grain Size

QEMSCAN analysis was used to determine the awaruite and magnetite grain size distributions. Distributions are conventionally determined by assuming the two-dimensional (2D) QEMSCAN images represent three-dimension (3D) spherical particles. This analytical method was used for magnetite, but not for awaruite where non-spherical awaruite particles (typically rods or flakes) are typically observed due to the malleability of awaruite. Instead awaruite grain size distributions were calculated based on the 2D QEMSCAN images as corrections to 3D particles could not be confidently applied.

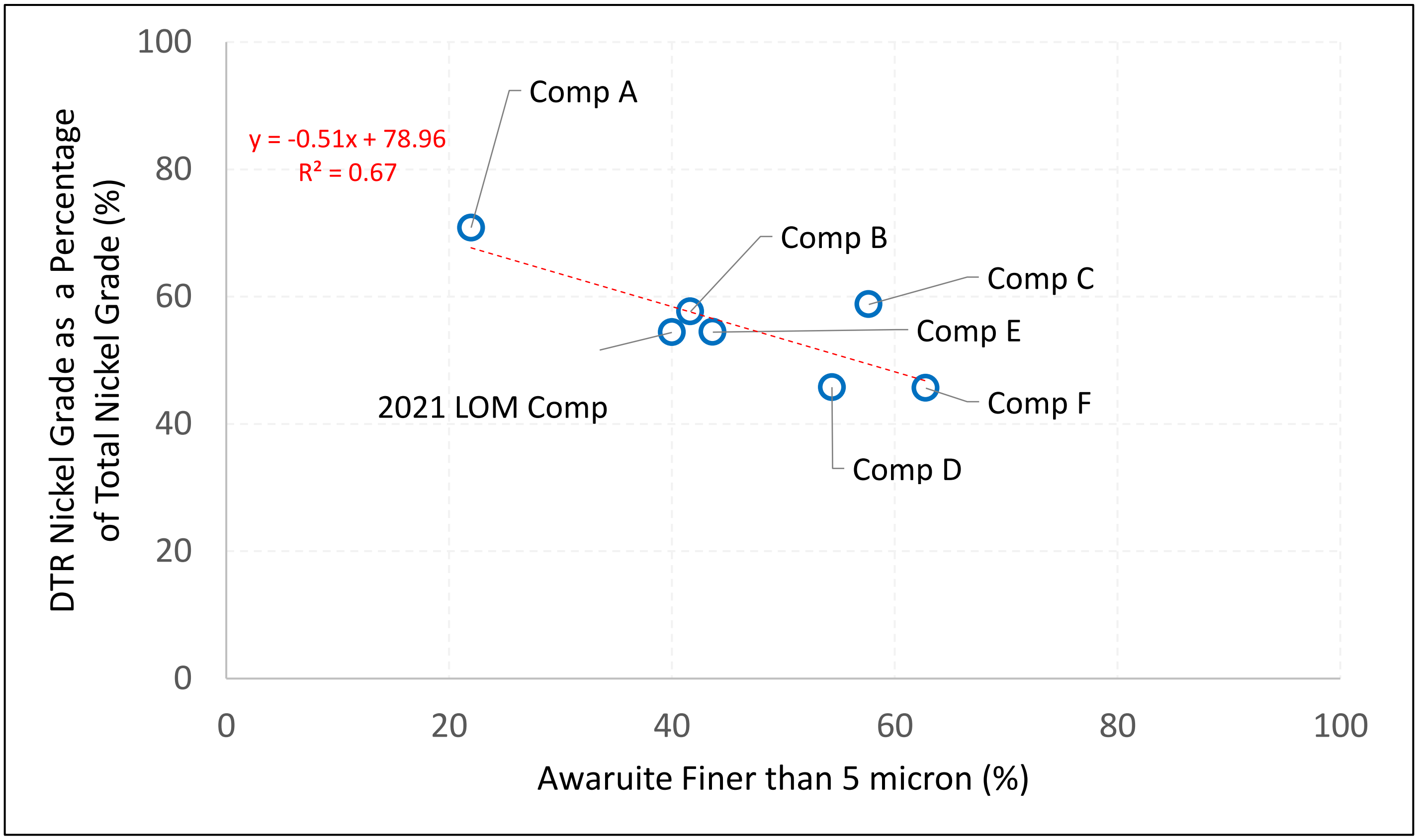
The awaruite grain size distributions are presented in Figure 13-13. Awaruite grains size distributions vary throughout the Deposit, with the early years mine phases showing the coarsest distribution and the later years the finest. A trend between total nickel reporting as DTR Ni and the proportion of very fine grained awaruite (<5 µm) was observed and is demonstrated in Figure 13-14. This is expected as very fine awaruite grains would not be sufficiently liberated at the DTR pulverization size (95% passing 75 µm) to be recovered. Even liberated very fine awaruite grains could potentially have insufficient mass for the magnetic field to overcome hydraulic drag in the Davis tube. The relationship between awaruite grain size and awaruite not reporting to DTR Ni is also presented in the note included in Figure 13-13.

Figure 13-13: Awaruite Grain Size Distribution (2D Analysis)



Source: FPX, 2023.

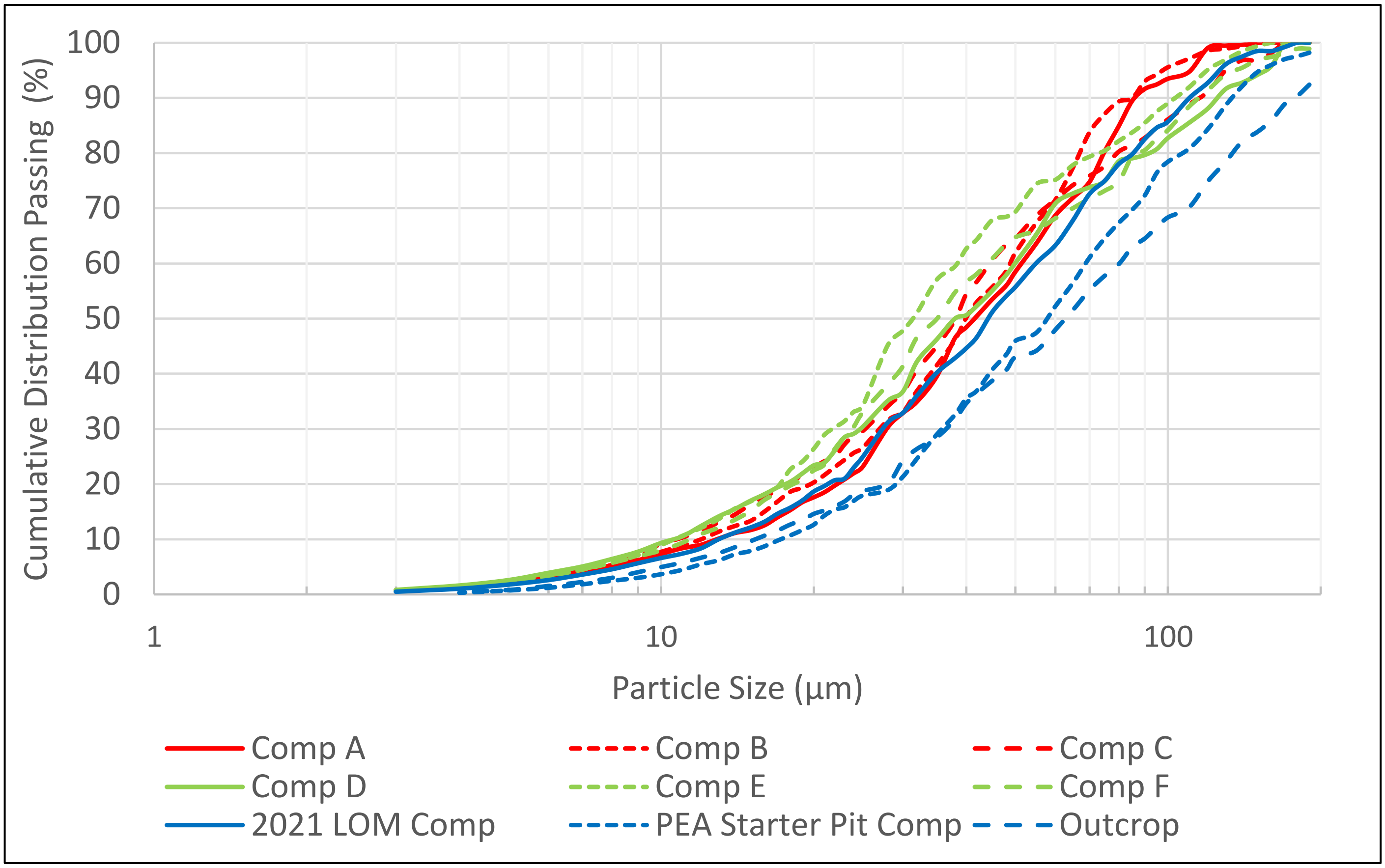
Figure 13-14: Relationship Between Total Nickel Reporting as DTR Ni and Proportion of Very Fine-Grained Awaruite



Source: FPX, 2023.

The grain size distributions for magnetite are summarized in Figure 13-15. Relative to awaruite, the magnetite grain size distributions show lower variability across the mine phases. The correlation between magnetite content and DTR Fe (Figure 13-12) and the consistency of magnetite grain sizes across the deposit suggest that DTR Fe could be a useful geometallurgical tool to predict mass pull in primary and regrind magnetic separation. No compelling relationship between magnetite grain size and awaruite grain size was observed.

Figure 13-15: Magnetite Grain Size Distribution (3D analysis)



Source: FPX, 2023.

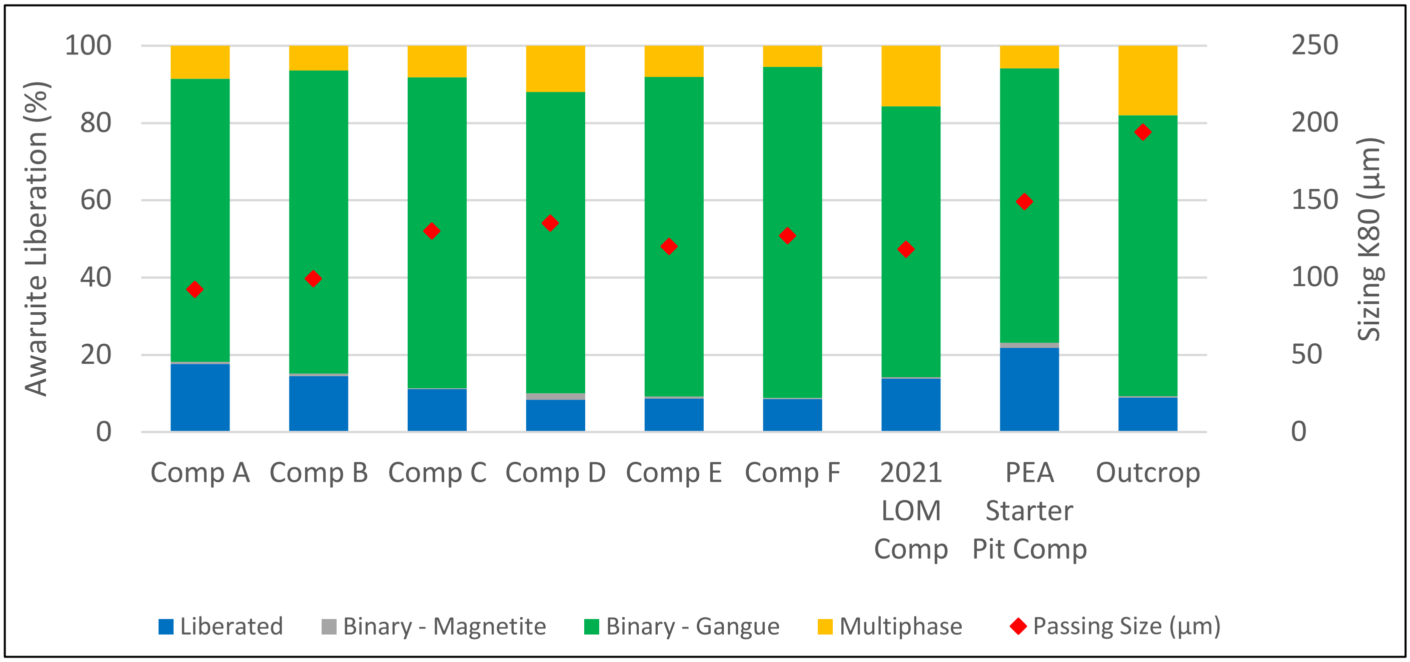
13.4.4 Mineral Liberation

QEMSCAN analysis was used to determine the awaruite and magnetite liberation extents and associations. Figure 13-16 and Figure 13-17 shows the liberation extent and association of awaruite and magnetite, respectively. The 80% passing particles size of the analyzed samples are overlaid on these charts and range from 92 µm to 194 µm. 9-22% of awaruite and 11-20% of magnetite was liberated at these sizings. This demonstrates that particle sizing was not the main driver of awaruite liberation with significantly higher awaruite liberation being recorded for a relatively coarser sample (PEA Starter Pit Comp), as compared to other composites at finer sizings. In contrast, magnetite liberations appeared to be driven by grind sizing as coarser samples generally measured lower magnetite liberations than samples at finer sizings.

Figure 13-18 and Figure 13-19 further detail the liberations of awaruite and magnetite and show the fraction of liberated minerals by size fraction. With the exception of the PEA starter pit composite, the majority of liberated awaruite exists in the minus 38 µm size fraction. This is consistent with the conclusions from previous studies and supports the use of a final regrind size finer than 38 µm. The lack of liberation at coarser size fractions does not preclude the use of primary magnetic separation so long as the mass of magnetic mineral in a composite particle provides sufficient magnetic force in a given magnetic field to recover the composite particle. For magnetite, a significant majority of liberated particles exist in the minus 38 µm size fraction. Liberation of magnetite in the final magnetic concentrate is important to reduce the content of acid-consuming magnesium minerals in the flotation feed.

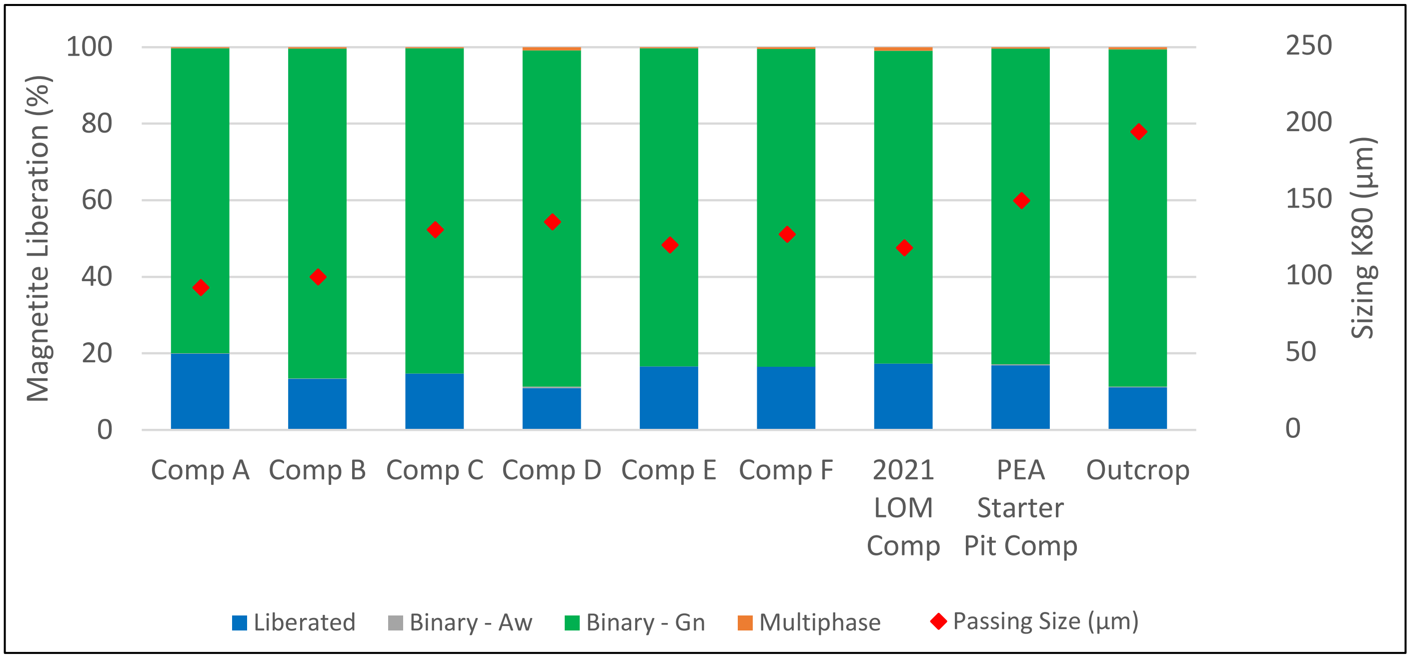
In this portion of the QEMSCAN analysis, the awaruite grouping includes both awaruite and nickel sulphides and therefore no association between these minerals is reported. However, historic investigations by Cliffs using optical microscopy identified a pentlandite association with awaruite. It is this association that is theorized to be the source of minor sulphur transfer to the flotation concentrate.

Figure 13-16: Awaruite Liberation by Class of Association



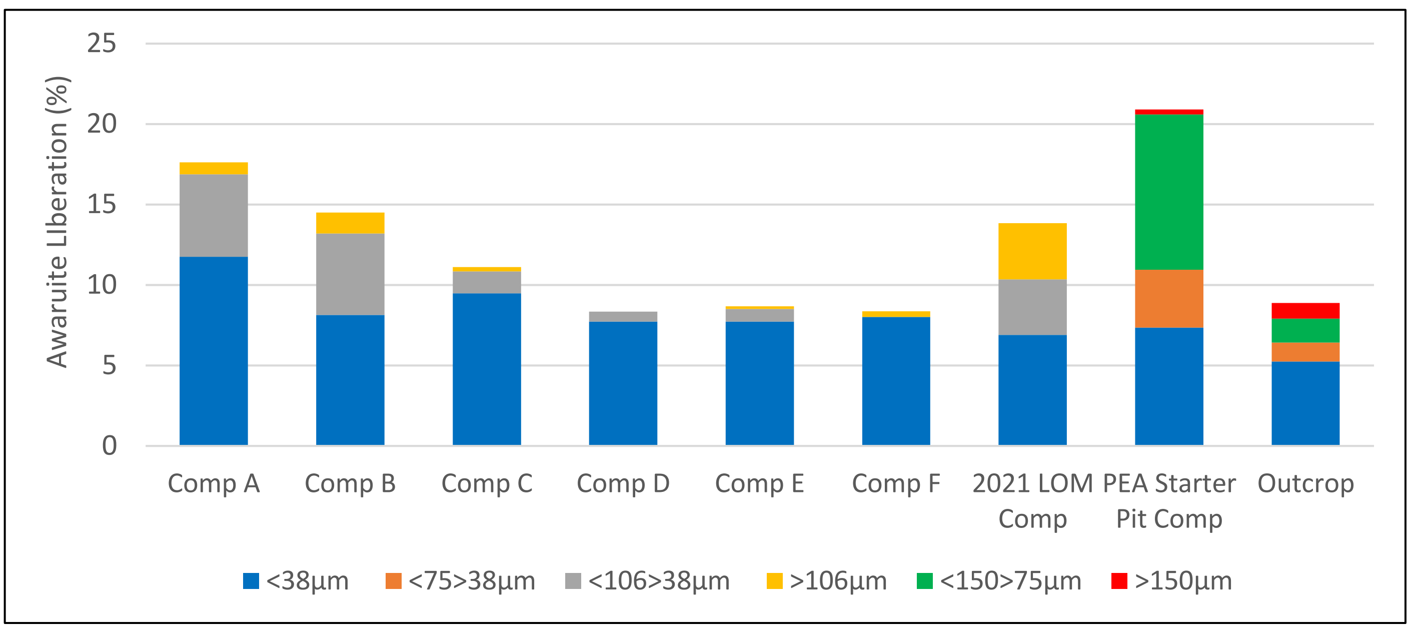
Source: FPX, 2023.

Figure 13-17: Magnetite Liberation by Class of Association



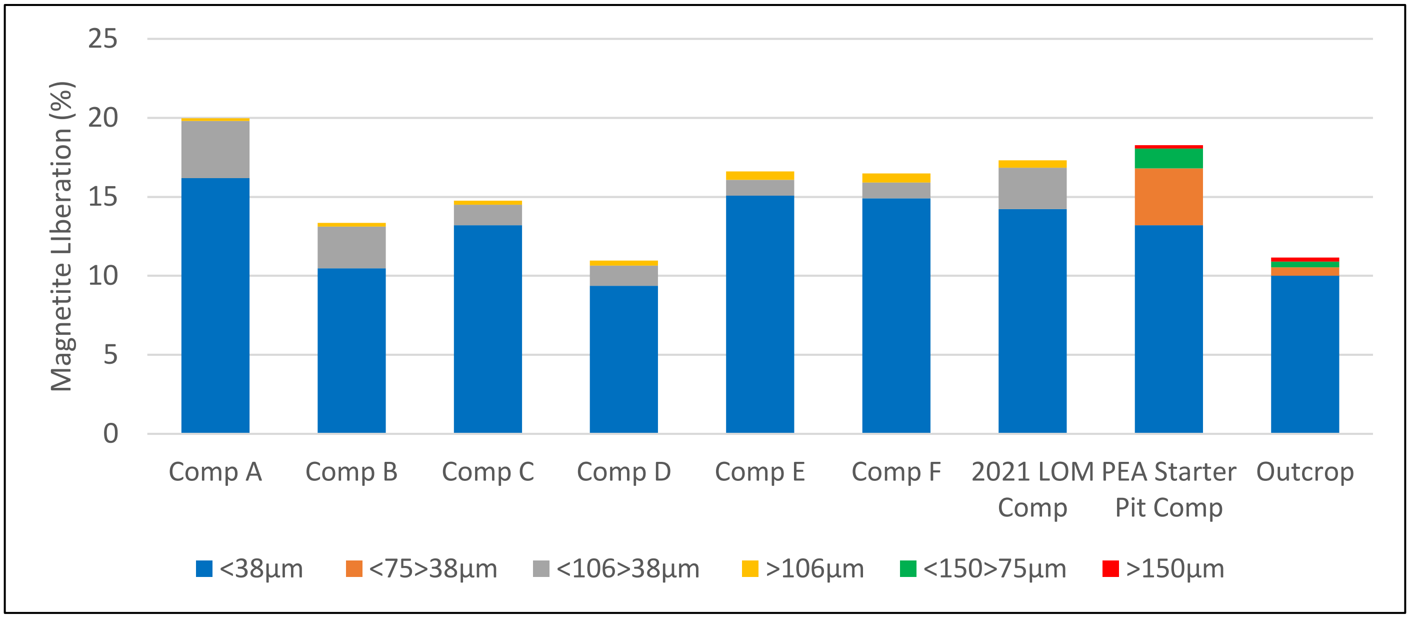
Source: FPX, 2023.

Figure 13-18: Liberated Awaruite by Sizing Fraction



Source: FPX, 2023.

Figure 13-19: Liberated Magnetite by Sizing Fraction



Source: FPX, 2023.

13.5 Comminution

The comminution design selected for the 2020 PEA was based on limited comminution characterization test data and utilized high-pressure grinding rolls (HPGR). Additional comminution characterization testing was completed for the PFS to assess the amenability of Baptiste ore to both HPGR technology as well as more conventional semi-autogenous grinding (SAG) mill technology. Both the HPGR and SAG circuits are preceded by primary crushing (and secondary crushing in the HPGR circuit) and proceeded by ball milling for final size reduction ahead of primary magnetic separation.

This comminution characterization data fed into an engineering trade-off study to assess the HPGR- and SAG-based circuits. This study concluded with the recommendation of the use of a SAG-based circuit which has been incorporated into the PFS engineering design. At a high level, the characterization testing highlighted that Baptiste ore is readily amenable to SAG milling; however, the Baptiste ore exhibited unusual behavior in HPGR testwork, related to the serpentine mineralization, which resulted in relatively low m-dot values and a resultant increase in HPGR equipment requirements. As HPGR is no longer part of the process plant design, further detailed commentary on the HPGR testing is excluded from this report.

Comminution characterization test work for the SAG circuit was conducted at ALS Metallurgy, SGS Lakefield, and Corem on the mine phase composites, the outcrop, and drill core 12HG04A. Samples were tested to assess the material competency, hardness, and abrasion characteristics. Samples, or a portion of samples, were submitted to the following characterization tests:

- JK drop weight (JKDW) test, or the Steve Morrell Comminution (SMC) test (the abbreviated version of the JKDW);
- Bond ball mill grindability test;

- Bond rod mill grindability test;
- MacPherson autogenous grindability test;
- Bond low-energy impact crusher test; and
- Bond abrasion index test.

Summary data and key observations from each series of tests are summarized in the proceeding sections.

13.5.1 JKDW / SMC Test

A total of 13 SMC (including 4 variability repeats) and 2 JKDW tests were completed on the mine phase composites (A-F, X, master comp) at ALS Metallurgy, SGS Lakefield and Corem and the resultant data was analyzed by JKTech and SMC Testing Pty LTD. Most SMC tests were performed at the -31+26.5mm size class range with a few at the -22.4+19mm size class range due to limited sample availability.

The Axb parameter (a measure of the material's resistance to impact breakage) ranged from 41-64 and can be characterized as moderately competent, consistent with other low grade nickel ultramafic deposits. A definite increase in impact resistance with decreasing particle size was observed. A review of the relationship between t_{10} and ECS from the JKDW tests shows a negligible spread of data between size classes, indicating a uniform breakage rate across the range of size classes.

A 75th percentile Axb value of 44.1 was selected for the PFS process plant design. The six mine phase composites (composite A-F) demonstrated limited variability with an average of 44.8 and a standard deviation of 2.5.

13.5.2 Bond Ball Mill Grindability Test

A total of 15 ball mill tests (including 5 variability repeats) were completed at ALS Metallurgy, SGS Lakefield and Corem on all samples used in the comminution program. The Bond ball work index (BWi) ranged from 19.3-24.8 kWh/t and is characterized as very hard in terms of their BWi. The six mine phase composites (composite A-F), tested at a single laboratory, demonstrated limited variability with an average of 21.2 and a standard deviation of 1.3.

These results include repeats conducted at different labs which returned a difference of 16%. As this range is greater than typically expected a quality control check on the results of the two test facilities was conducted using the Levin B value. Both data sets passed the quality control check, but one at the upper end of the acceptable range and the other at the bottom end of the acceptable range.

The highest BWi value, 24.8 was returned from composite X, which was expected to be comparable to composite A/B (BWi 19.6/20.9). A review of composite X intervals core photos indicated isolated lenses of dunite were captured in the composite. Composite C and D also have elevated levels of dunite, but only reported slightly elevated BWI values (BWi 22.5/22.9). Further, the mine phase master composite returned a BWi 13% higher than the weighted average BWi of its constituent components suggesting potential greater sample variability than captured in the mine phase composite A-F results. Further comminution characterization testing on discreet drill core intervals on different rock types, including dunite and any other notably harder rock types, is recommended during the feasibility stage.

Most BWi results were all conducted at the traditional BWi closing size of 100 mesh (150 µm) and returned a product 80% passing size around 115 µm. Three additional BWi tests (composites C, D, and E) were conducted at a closing size of 40 mesh (420 µm) and returned a product 80% passing size around 275 µm, which is much closer aligned to the selected process plant design P_{80} of 250 µm. The tests at the coarse closing size on average has a BWi 13% higher than that of the smaller closing size. An increase in hardness at coarser grind sizes is not unexpected, though the magnitude of the difference is noteworthy.

Given the discussed uncertainty in BWi value, the largest BWi recorded at the coarser closing size, BWi = 23.5, was selected for the PFS process plant design.

13.5.3 Bond Rod Mill Grindability Test

A total of seven rod mill tests (including one variability repeat) were completed at ALS Metallurgy, SGS Lakefield and Corem on mine phase composites C/D/E, the mine phase master composite, and the outcrop. The Bond Rod Mill Work Index (RWi) ranged from 15.7-18.3 kWh/t and is characterized as hard in terms of their RWi. The ratio between RWi and BWi ranged from 0.71-0.95. A RWi index of 17.6 was selected for the PFS process plant design.

13.5.4 MacPherson Autogenous Grindability Test

Across various mineral deposits, the Axb and RWi values typically demonstrate a relationship that falls into a known range. The Baptiste samples fell outside of this range and this disagreement was enough that mills designed with a Bond-type model will be materially larger than mills designed using a Morrell (SMC) type model. This suggests that there is a Baptiste material characteristic that is confusing one (or both) of the bench-scale tests. To further investigate and clarify this phenomena, Macpherson autogenous grindability tests were completed and the results of which were expected to serve as a third/umpire testing between mill sizing methods.

Three MacPherson tests were completed at SGS Lakefield on mine phase composites C, D, and E. The three samples returned autogenous work index (Awi) values ranging between 9.3-9.5 kWh/t characterizing them as moderately hard with respect to the Awi. None of the three tests indicated a build-up of critical size material in the mill charge.

The McPherson tests corroborated the Axb parameter from the SMC Test. It is therefore recommended that a Morrell (SMC) type model be used for mill sizing and not the Bond-type model. The PFS mill sizing, conducted with Ausenco's in-house model (Ausgrind) is consistent with this approach.

13.5.5 Bond Low-Energy Impact Crusher Test

Crusher tests were completed on two samples. The 12HG04A and outcrop samples reported a CWi of 4.3 and 11.7 kWh/t respectively. The 12HG04A crusher work index (CWi) appeared very low and inconsistent based on the expected Axb and CWi relationship. The 12HG04A sample is located on the periphery of Phase 1 near faults and had some patchy incipient Fe-carbonate alteration. These results were not considered in the plant design and a CWi of 11.7 was used for PFS process plant design.

13.5.6 Bond Abrasion Test

A total of five abrasion tests (including two variability repeats) were completed at ALS Metallurgy and Corem on mine phase composite X, the mine phase master composite, and the outcrop. Abrasion index (Ai) values range from 0.007-0.010 characterizing them as extremely low abrasivity and is typical of other low-grade nickel ultramafic deposits. A design value of 0.010 was selected for PFS process plant grinding media consumption rates.

13.5.7 Rock Quality Designation

Geotechnical features of the Baptiste material were also considered for the design of the SAG circuit. The interpretation of the Rock Quality Designation (RQD) data allows capturing the effects that rock fractures have on crushing performance and on the SAG mill feed size (F_{80}), that in turn impacts the SAG mill specific energy.

The average RQD value of all logged core intervals is approximately 50%, indicating a fair quality for overall rock mass within the Baptiste Deposit. To account for the possibility that the RQD values are typically under reported due to exploration drilling and handling methods, a higher RQD value of 60% was selected for the PFS process plant design. This RQD design value yields a resultant SAG feed P_{80} of 85 mm based a ROM P_{80} of 415 mm, and a 120 mm closed side setting on the primary crusher. Intensive mine blasting practices are required to achieve the target ROM feed size distribution.

RQD data by mine phase is summarized in Table 13-6 and shows RQD is skewed toward lower RQD values in earlier mine phases. A histogram of life-of-mine RQD was previously presented in Section 13.3.1, Figure 13-14.

Table 13-6: Summary of RQD Statistics by Mine Phase

Mine Phase	1	2	3	4	5	6	7	8	LOM
Average RQD (%)	39	40	48	41	48	39	45	51	43
Percentage < RQD 50 (%)	67	74	63	67	59	80	65	61	61

13.6 Magnetic Separation

13.6.1 Introduction

The key variables investigated during the PFS metallurgical testing program were primary grind size, the impact of preferential grinding, and magnetic field strength. The key responses to these variables are the DTR Ni recovery and mass pull.

The 2020 PEA envisioned the use of a tailings facility partially constructed out of cyclone tailings sands. This restricted the primary grind size which was optimized for sand production (at P_{80} = 300 µm) rather than DTR Ni recovery. As the PFS envisions rock-fill embankments in the tailings facility, thus this former grind size restriction is alleviated. To define the best value combination of recovery and grind size, the impact of preferential grinding, and increased magnetic field strengths were investigated.

The high specific gravity of awaruite (SG = 8.6) compared to the background gangue minerals (SG ~2.6) was expected to result in preferential grinding of awaruite in a milling circuit operating in closed circuit with a hydrocyclone classifier, which classifies minerals based both on particle size and particle density. The same was expected, to a lesser extent, of magnetite (SG = 5.2). This preferential grinding (and liberation) of awaruite was suspected to allow the use of a coarse grind size while maintaining high levels of DTR Ni recovery. As bench-scale grinding is conducted using screens for classification (which do not classify on the basis of SG), pilot-scale grinding using a hydrocyclone was investigated in the PFS.

Increased magnetic field strengths were expected to increase the recovery of fine or partially liberated magnetic particles. Only relative minor field strength increases were considered, up to a 2,000 G (as measured on the drum working surface). This field strength typically can be generated in industrial equipment using ferrite/ceramic magnets, rather than more costly rare-earth magnets. The cut-off between low intensity and medium intensity magnetic separation has historically been 2,000 G for this reason. The PFS process plant design uses low intensity wet magnetic separation drum separators (same as the PEA design).

13.6.2 Primary Magnetic Separation

13.6.2.1 2020 PEA Bench Testing

Early investigations into the relationship between grind size and mass pull versus nickel recovery are summarized in Figure 13-20 and Figure 13-21, respectively. The ALS testing on the PEA starter pit composite using a bench-scale LIMS at 1,100G shows a clear decrease in recovery with increasing grind size. However, the SGS Lakefield testing on a historic composite using a Davis tube at 3,500G showed a parabolic relationship with recovery first dropping, then increasing. This increase in recovery at coarser size is due to increasing mass pull, which pulls increasing amounts of non-magnetic, unliberated nickel. The increase in nickel recovery is misleading as a significant portion of this non-magnetic nickel would then be rejected in magnetic separation cleaning stages following regrind. All PFS test work was evaluated on a DTR Ni basis to remove this potential source of misinterpretation.

Figure 13-20: Relationship Between Primary Grind Size and Total Nickel Recovery

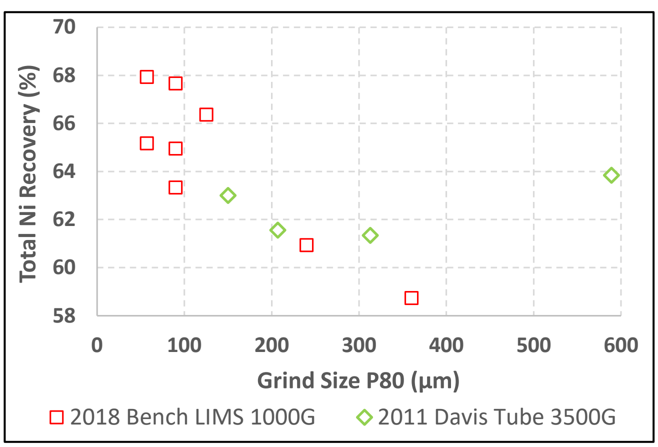
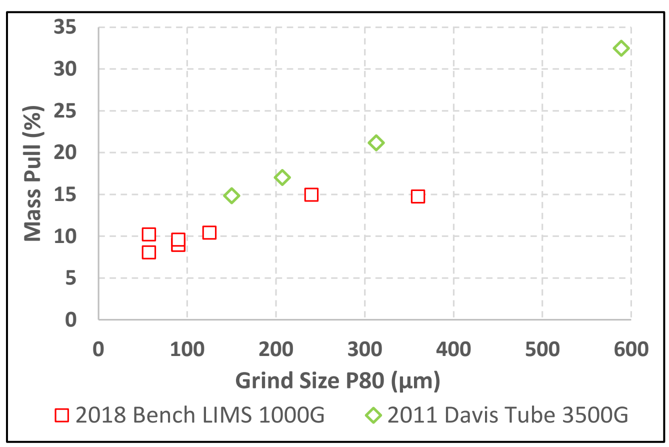


Figure 13-21: Relationship Between Primary Grind Size and Mass Pull



Source: FPX, 2023.

13.6.2.2 PFS Bench Testing

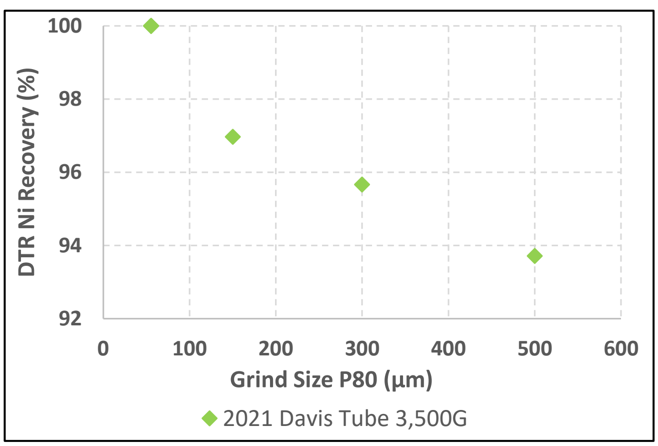
Figure 13-22 and Figure 13-23 show the impact of grind size on DTR Ni recovery and mass pull on the 2021 LOM composite using a Davis tube at 3,500 G. In contrast to the PEA testing which measured total nickel recovery, clear DTR Ni and mass pull relationships with grind size were observed. Note that by definition, the test at 55 µm has 100% DTR recovery, as these test conditions were near identical to the DTR procedure conditions of 95% passing 75 µm.

To isolate the impact of magnetic field strength, a series of bench-scale LIMS tests were conducted using the outcrop material milled in the pilot plant to a P_{80} of 280 µm. Figure 13-24 and Figure 13-25 summarizes the results which demonstrate a clear relationship between increasing field strength and both DTR Ni recovery and mass pull.

Figure 13-26 compares the mass pull and DTR Ni recovery demonstrating that DTR Ni recovery beyond 90% is nearly linear with increasing magnetic field strength. Of note is the DTR Ni recovery of 98% at around 32% mass pull, indicating successful liberation of DTR Ni from the remaining 68% mass reporting to tailings.

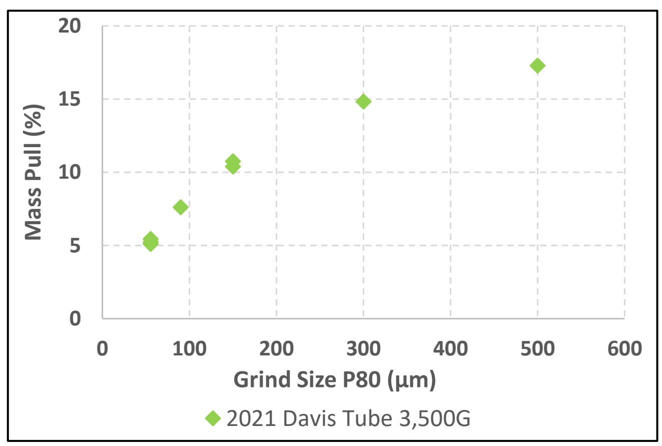
A final observation from the magnetic field strength testing was noted when comparing the DTR Ni and DTR Fe recoveries as presented in Figure 13-27. This suggests that DTR Ni and DTR Fe recoveries are similar when the combination of grind size and magnetic field strength are sufficient to produce high recoveries. As grind size increases and/or magnetic field strength decreases, the loss in DTR Fe recovery exceeds the loss in DTR Ni recovery. This is expected given the significantly higher magnetic susceptibility (10x) of awaruite compared to magnetite. The increasing DTR Fe recovery at higher field strengths were accompanied by a decreasing DTR Fe grade in the magnetite concentrate indicating the incremental recovery was from middling particles associated with gangue.

Figure 13-22: Relationship Between Primary Grind Size and DTR Ni Recovery



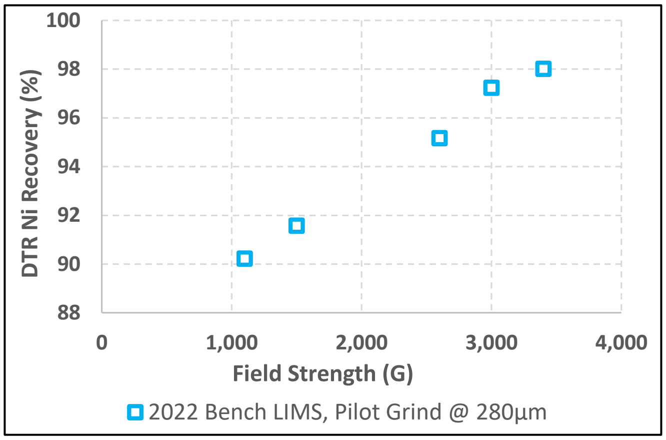
Source: FPX, 2023.

Figure 13-23: Relationship Between Primary Grind Size and Mass Pull



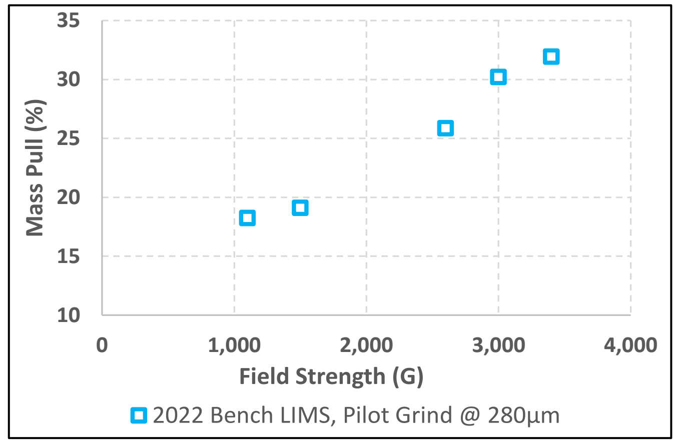
Source: FPX, 2023.

Figure 13-24: Relationship Between Magnetic Field Strength and DTR Ni Recovery



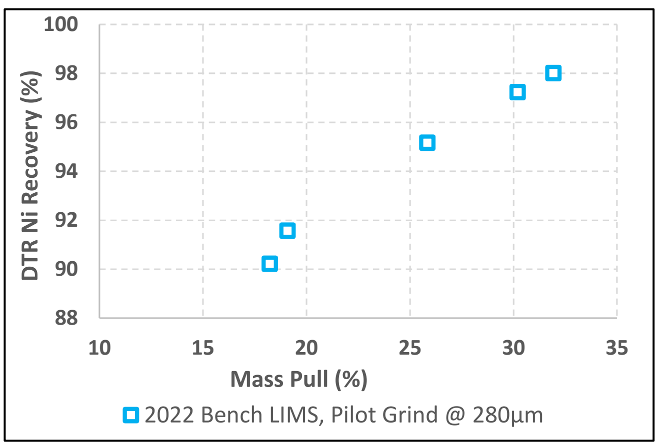
Source: FPX, 2023.

Figure 13-25: Relationship Between Magnetic Field Strength and Mass Pull



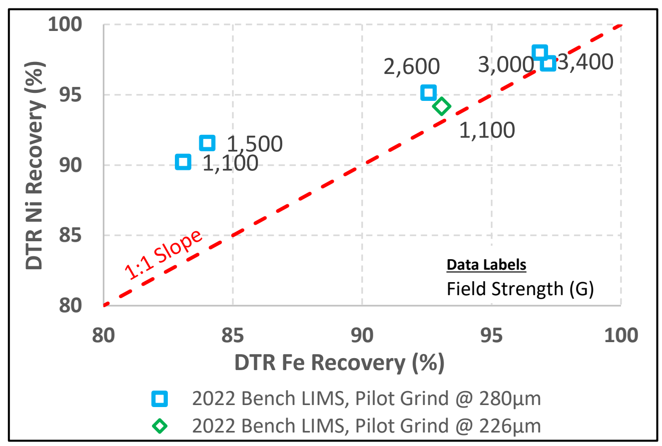
Source: FPX, 2023.

Figure 13-26: Relationship Between Mass Pull and DTR Ni Recovery



Source: FPX, 2023.

Figure 13-27: Impact of Grind Size and Magnetic Field Strength on the Relationship Between DTR Ni and DTR Fe Recovery



Source: FPX, 2023.

13.6.2.3 PFS Pilot Plant Testing

Two magnetic separation pilot plants were completed during the PFS. The first pilot plant was completed at SGS Lakefield using the 2021 LOM composite and had the primary objectives of:

- generating a substantial quantity of magnetic concentrate to serve as flotation feed for a detailed flotation program to demonstrate final upgrading of the awaruite concentrate; and
- investigating the impact of preferential grinding on magnetic separation performance.

The second pilot plant was completed at Corem using the outcrop and had the primary objectives of:

- generating a substantial quantity of awaruite concentrate to serve as feed for a detailed hydrometallurgy refining program to produce battery-grade nickel sulphate;

- providing further insights into the benefits of preferential grinding; and
- generating various magnetic separation intermediate product material for bench-scale optimization testing.

A notable limitation of both pilot plants is that, due to flow rate and cyclone sizing limitations, the cyclone feed pressures generated in the pilot plant are significantly lower than those expected in commercial operation. The lower cyclone feed pressures in the pilot plant testing is expected to have reduced the efficacy of the hydrocyclone, and the results likely underreport the preferential grinding benefit.

This section discusses only the primary grinding and magnetic separation portions of the pilot plants. The regrind and cleaner magnetic separation portions are discussed separately in Section 13.6.3.

13.6.2.3.1 2021 Pilot Plant

The 2021 pilot plant, consisting of grinding and magnetic separation, was completed at SGS Lakefield using the 2021 LOM composite. The pilot plant processed at a total of 3.55 tonnes of material at a primary grind size P_{80} ranging between 116-256 µm. The average primary grind throughput rate was 157 kg/h for a total operating time of 23 hours. Primary grinding runs were run in closed circuit with either a hydrocyclone or a screen. Magnetic separation was conducted using a rougher scavenger LIMS configuration at field strength of 1,000 G.

When the cyclone was in operation, a large build-up of nickel and iron was measured in the cyclone underflow stream, as expected. Size-by-size analysis of that stream showed preferential grinding of the nickel. However, the impact of this preferential grinding on magnetic separation performance was not conclusively demonstrated due to metallurgical balancing issuing across the milling operation.

In the limited time this pilot plant was operated, a portion of the nickel and some of the iron that built up around the classification and grinding circuit never reached the cyclone overflow and the downstream LIMS separation circuit. This caused a decrease in DTR Ni grade to the rougher LIMS feed of more than 20% compared to the fresh mill feed head grade. When the screen classification replaced the cyclone, the LIMS feed grade was similar to the fresh mill feed head grade. At the completion of the milling run an attempt was made to recovery all nickel in the mill, however these efforts were not able to reconcile the low nickel grades in the cyclone overflow. Two key learnings from this were that a) longer operational durations where required to permit a build-up of awaruite in the mill circulating load and b) a greater emphasis on mill housekeeping was required as even a small mill spill can represent a material quantity of DTR Ni due to the high circulating load of nickel.

Although the impact of preferential grinding could not be conclusively demonstrated, the non-mags DTR Ni assay do provide some insights. When the cyclone was in operation, the nickel grade in the scavenger LIMS tailings was approximately 10% lower than when the screen was in operation. While this could potentially indicate a preferential grinding benefit, the results are confounded by the lower LIMS feed grade during the cyclone run. Because not all the nickel could be accounted for when the cyclone was in operation, it is difficult to compare the performance of the two modes of operation. It is also difficult to predict what could have been the nickel grade in the scavenger tailings if all the nickel would have reported to the cyclone overflow. It is likely that the preferential grinding measured in the cyclone underflow increased the awaruite liberation and could have in turn increased selectivity and awaruite recovery in the LIMS separation. However, because only ~60% of the nickel in the cyclone underflow was found in the form of DTR Ni, the liberation of non-recoverable awaruite and other nickel mineral carriers has also likely increased, possibly increasing the scavenger tailings grade if it reported to the cyclone overflow and LIMS circuit.

Based on the encouraging reductions in scavenger LIMS tailings grades observed in the 2021 pilot plant, a subsequent pilot plant was run in 2022 to conclusively define any preferential grinding benefit. Based on the learnings from the 2021 pilot plant, the 2022 pilot plant was operated for a longer operational duration with a greater emphasis of mill spill management.

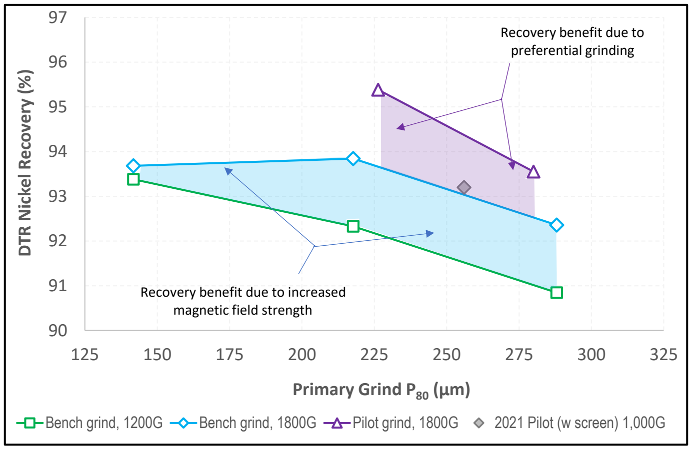
13.6.2.3.2 2022 Pilot Plant

The 2022 pilot plant, consisting of grinding and magnetic separation, was completed at Corem using bulk outcrop material. The pilot plant processed a total of 16.3 t of material at a primary grind size P_{80} ranging between 146-280 µm. The average primary grind throughput rate was 207 kg/h with a total operating time of 79 hours. Primary grinding runs were run in closed circuit with a hydrocyclone. Magnetic separation was conducted using a rougher scavenger LIMS configuration at field strength of 1,200 G. A second scavenger stage was also evaluated at a higher magnetic field strength of 1,800-2,200 G. As the incremental DTR Ni recovery was minimal in the first scavenger, typically 1-2% of the rougher recovery, the additional recovery observed in the second scavenger was attributed solely to the increased field strength rather than an additional scavenging stage.

To further evaluate the pilot plant data, a series of tests were run with batch ground material. These tests were run with the same pilot-scale magnetic separation equipment with the goal of isolating the impact of preferential grinding.

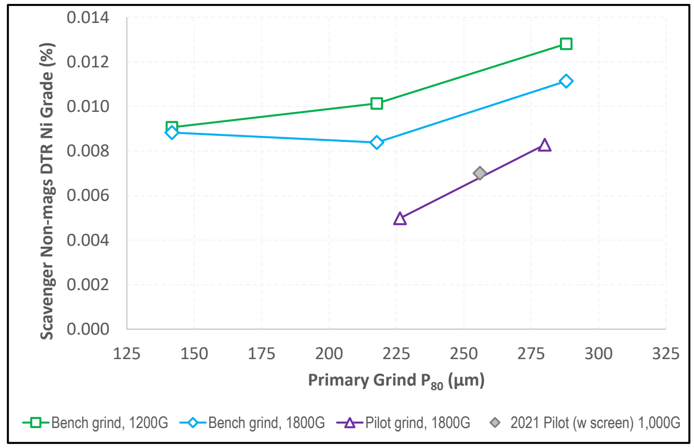
Figure 13-28 shows a clear recovery benefit of increased field strength and preferential grinding on DTR Ni recoveries at coarser grind sizes, but this benefit is reduced at finer grind sizes. Figure 13-29 shows the same impact is observed when reviewing the impact on tailings grade. A recovery improvement of 1.5% between bench-scale testing and pilot-scale testing with preferential grinding is observed. For comparison purposes, the performance of the 2021 Pilot Plant when running at steady state with a screen and without an elevated field strength is also presented in these figures. The 2021 Pilot Plant run with a hydrocyclone is not presented as the circuit did not achieve steady state as the cyclone overflow grades were lower than mill feed grade (as discussed in Section 13.6.2.3.1).

Figure 13-28: Relationship Between Grind Size, Field Strength, and Preferential Grinding on DTR Ni Recovery



Source: FPX, 2023.

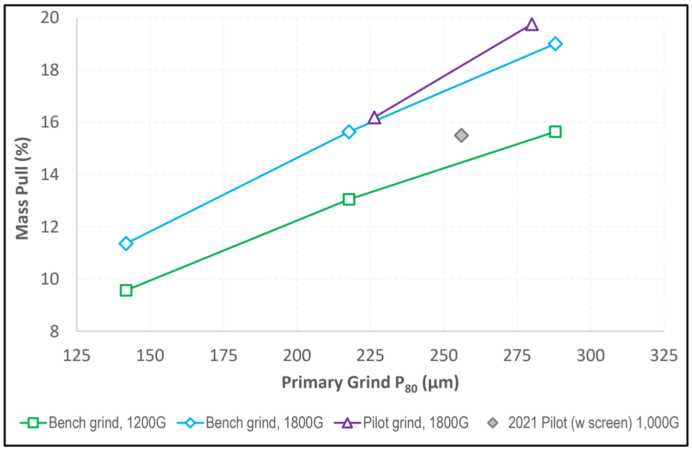
Figure 13-29: Relationship Between Grind Size, Field Strength, and Preferential Grinding on Tailings Grades



Source: FPX, 2023.

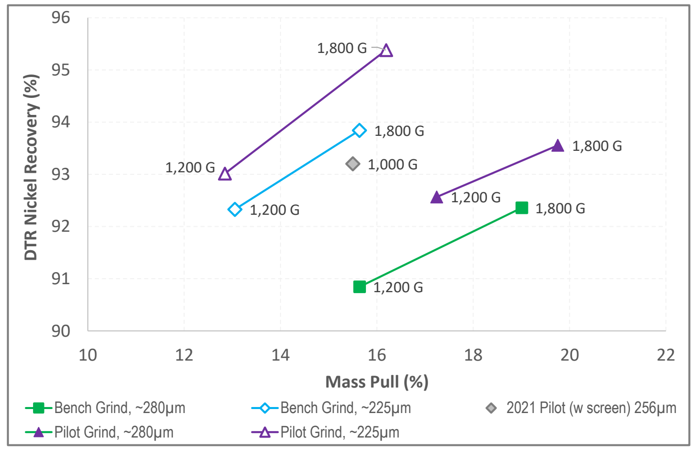
Figure 13-30 presents the relationship of increased field strength and preferential grinding on DTR Ni recoveries. This demonstrates that the increased recovery associated with increased field strength is accompanied by increased mass pull, as previously demonstrated (Figure 13-24). In contrast, the increased recovery associated with preferential grinding is not accompanied by materially increased mass pull. This relationship is further demonstrated in Figure 13-31 which shows the recovery and mass pull relationship. The shift of recovery from bench grinding to pilot grinding clearly demonstrates the impact of preferential grinding on the grade-recovery relationship. However, the extent of this shift is less than observed when reducing grind size from 280 µm to 225 µm. Based on Figure 13-31, it is estimated that preferential grinding provided the same benefit as reducing the grind size by 20-25 µm. The magnitude of this benefit shown is impacted by low hydrocyclone feed pressures achieved in the pilot plant, and operation at the higher hydrocyclone pressure typical of industrial-scale operation would reasonably be expected to provide further preferential grinding benefit.

Figure 13-30: Relationship Between Grind Size, Field Strength, and Preferential Grinding on Mass Pull



Source: FPX, 2023.

Figure 13-31: Relationship Between Mass Pull and DTR Ni Recovery



Source: FPX, 2023.

Compared to the 2022 work operating with a bench grind and low magnetic field strength and on the outcrop material, the 2021 LOM composite reports elevated DTR Ni recovery and lower scavenger tailings grades, albeit with an increased mass pull. Note that the 2021 LOM composite and the outcrop materials have similar DTR Fe feed grades which permit reasonable comparison of the mass pulls between different samples.

13.6.2.4 Variability

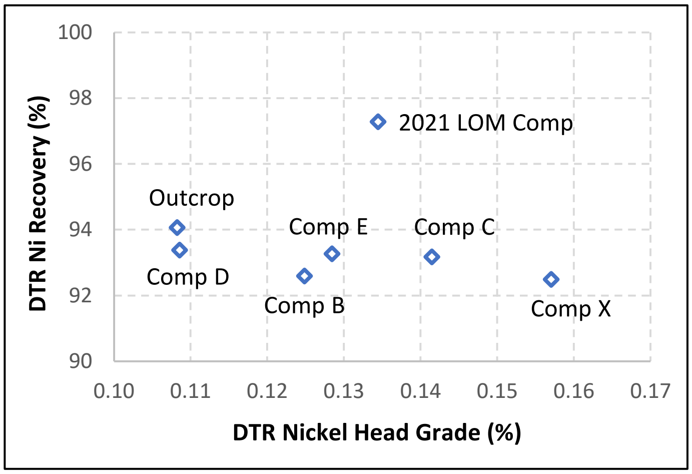
13.6.2.4.1 Variability Bench Testing

To assess the variability in magnetic separation response across the Baptiste Deposit, a variability program was conducted at Corem using the mine phase composites (composite B-F and X), the 2021 LOM composite, and the outcrop material. The variability program used a standardized testing protocol and included batch grinding to a P_{80} of 250 µm and rougher scavenger magnetic separation at 1,200 G and 1,800 G, respectively. The grind size and magnetic separation conditions are reflective of those selected for the PFS process plant design. The tests were conducted with a feed charge of approximately 20 kg. The outcrop sample was specifically tested to provide a baseline comparison between the variability test conditions, where no preferential grinding occurred, and the pilot plant conditions, where preferential grinding was observed.

The DTR Ni recovery response observed was relatively consistent with a median recovery of 93.5%. With one exception, all tests fell into the 93% ± 1% band. The 2021 LOM composite returned an anomalously high DTR Ni recovery of 97%. The high recovery is unexpected given as the LOM composite was expected to behave similarly to the average of the mine phase composite. No clear explanation for this result was determined. The DTR Ni feed grade ranged between 0.11 and 0.16% DTR Ni and no compelling grade-recovery relationship was observed. Composite F was excluded from this data analysis because a procedural error resulted in overgrinding with a resultantly higher DTR Ni recovery.

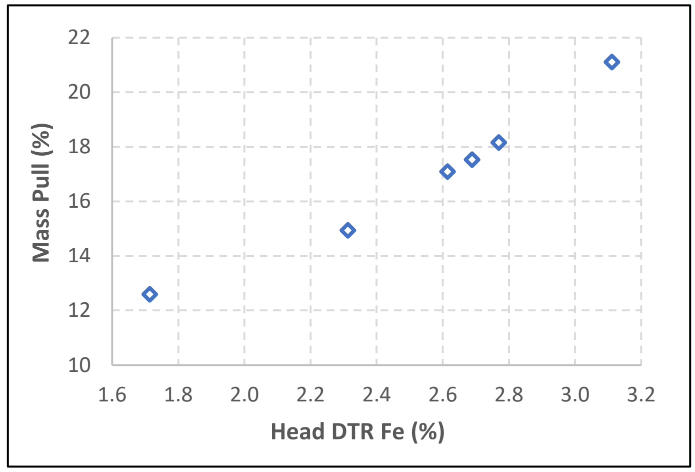
A clear relationship between DTR Fe head grade and mass pull was observed and is presented in Figure 13-33. This relationship is directly related to the relatively constant DTR Fe recovery around of 93% and magnetics DTR Fe grade of around 14%. This result was expected given the generally consistent grain size distribution on the magnetite identified during the mineralogy work (Figure 13-15). The relationship makes feed DTR Fe grade a useful geometallurgical tool for predicting primary magnetic separation mass pull.

Figure 13-32: Relationship Between Grind Size, Field Strength, and Preferential Grinding On Mass Pull



Source: FPX, 2023.

Figure 13-33: Relationship Between Head DTR Fe Grade and Mass Pull



Source: FPX, 2023.

13.6.2.4.2 Geological Database

As discussed in Section 13.1.2, the geological samples used to determine the resource model were analyzed on a DTR Ni basis using the Davis tube magnetic separator. The PFS mine plan encompasses over 7,000 discrete intervals that were all tested using the DTR methodology at spacing sufficient to support the resource estimate. As the DTR Ni recovery response observed in the variability testing is remarkably consistent, the DTR Ni data in the geological samples provides a granular estimate of magnetic separation DTR Ni recovery across the Baptiste Deposit. This estimate materially increases the confidence in the PFS magnetic separation recovery predictions and is especially valuable for nickel deposits where typically a significant and variable proportion of the nickel mineralization exists in a form that is unrecoverable through mineral processing.

Figure 13-34 and Figure 13-35 shows the cumulative distributions of the DTR Ni and DTR Fe assays, respectively, from all hole drill intervals contained within the PFS mine plan. Overlaid on the cumulative distribution is the samples tested as part of the variability bench testing, which indicates the 20th percentile and 80th percentile material has been tested. Note that the cumulative distribution is for discrete drill hole intervals; the distribution of plant feed grades would be narrower due to both the incidental and planned blending that occurs in mine operations.

Figure 13-34: Cumulative Distribution of DTR Ni Assays from Drill Hole Intervals within the PFS Mine Plan, Overlaid with Variability Samples Grades

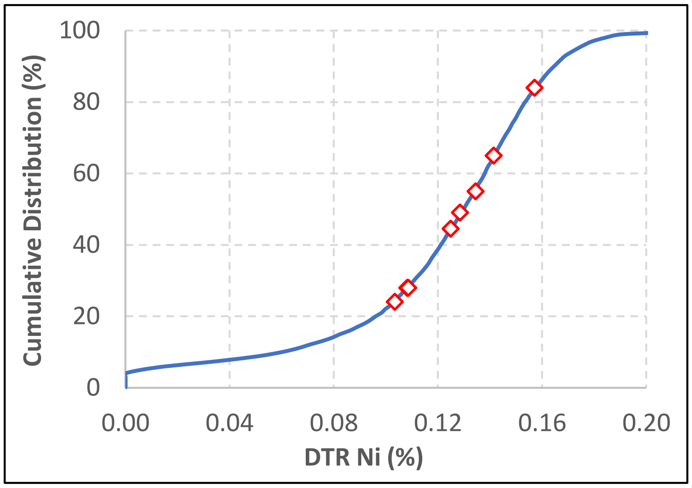
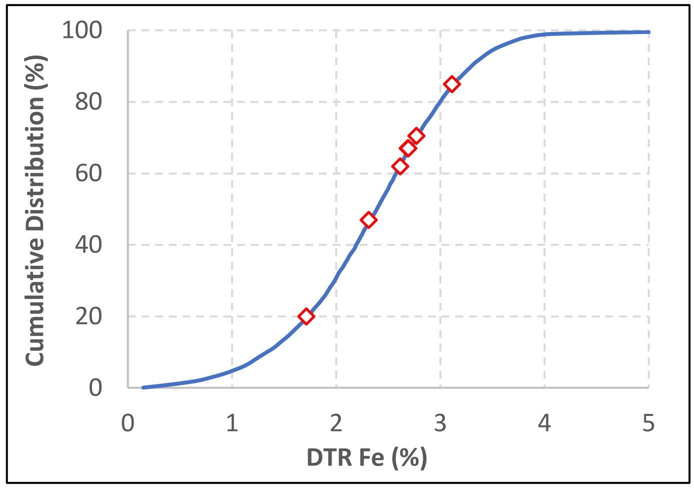


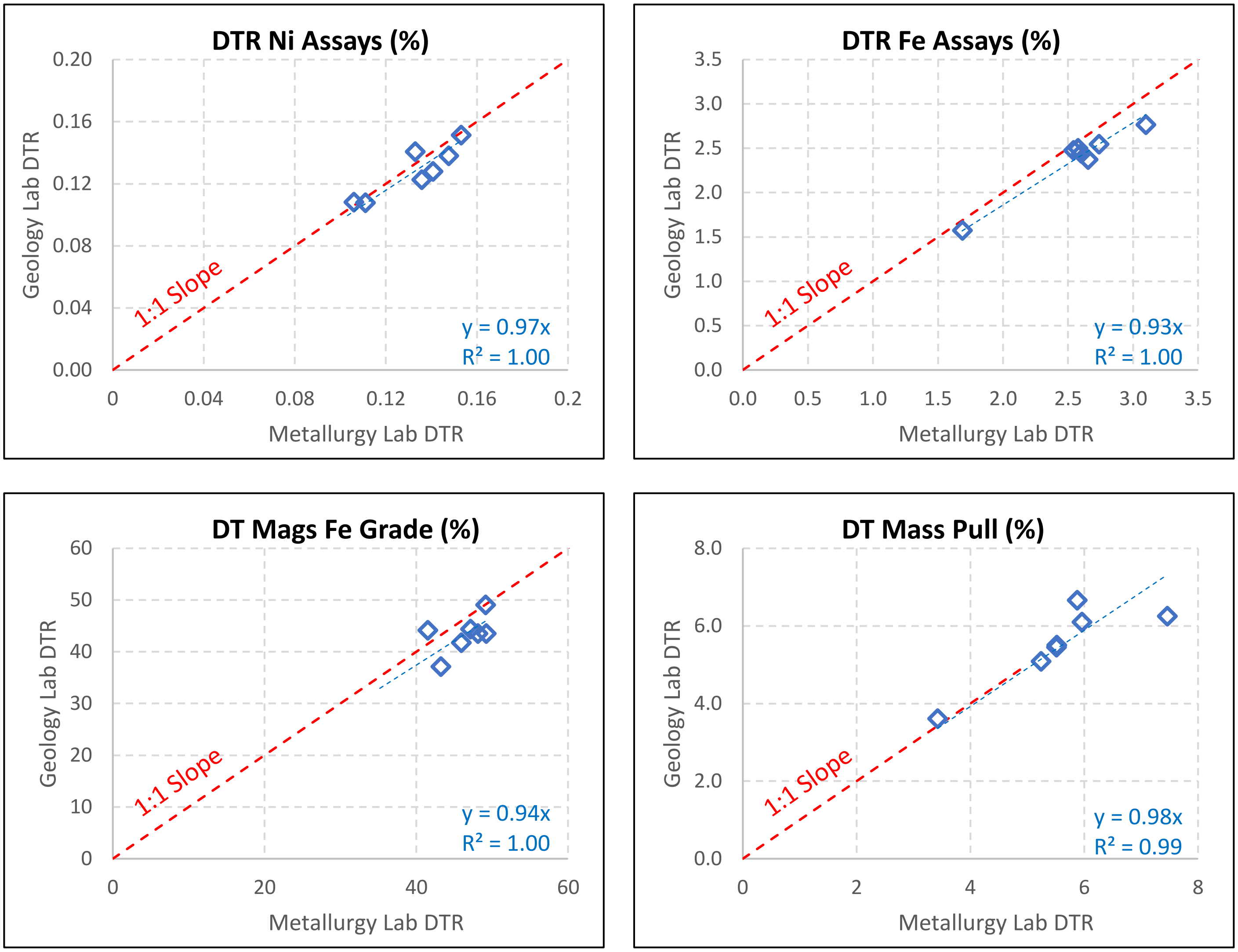
Figure 13-35: Cumulative Distribution of DTR Fe Assays from Drill Hole Intervals within The PFS Mine Plan, Overlaid with Variability Samples Grades



Source: FPX, 2023.

The DTR Ni recoveries found in the variability program were determined by the metallurgical laboratory (Corem) following a standardized procedure set by FPX. This standardized procedure was the same used for all geological assays used to build the resource model. To verify that the DTR procedure run by the metallurgical laboratory produced comparable results, the variability head samples were sent to the laboratory that analyzed the majority of the geological samples. A comparison of the two labs results in presented in Figure 13-36, which shows acceptable correlation. This supports the application of a DTR metallurgical recovery to the resource model.

Figure 13-36: Comparison of DTR Results Between Metallurgy Laboratory and Geology Laboratory



Source: FPX, 2023.

13.6.3 Cleaner Magnetic Separation

13.6.3.1 PEA Bench Testing

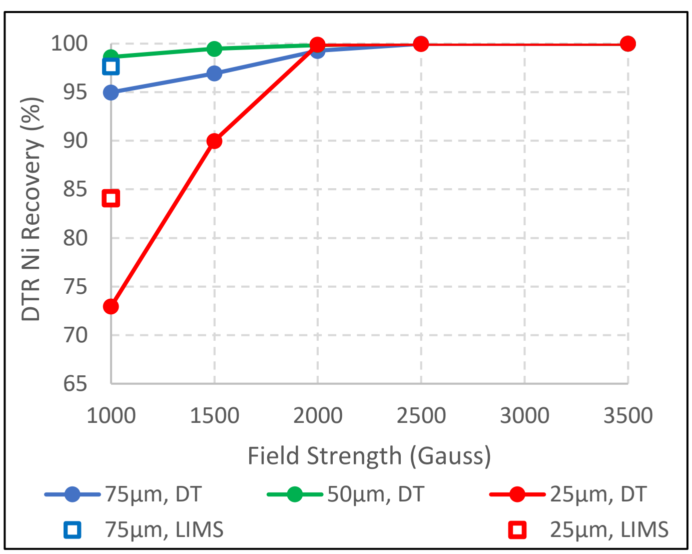
For the 2020 PEA metallurgical testing program a regrind size around 25 µm was generally applied. The total nickel recovery reported generally ranged between 85-90% in the cleaner stage. However, cleaner magnetic separation tests were conducted at a primary grind size finer than those envisioned for this study. This complicates DTR Ni recovery evaluations as coarser grinds are expected to carry more non-magnetic nickel that in turn is expected to be rejected during regrind and cleaning. While very high DTR Ni recoveries in cleaner magnetic separation can be reasonably expected, the absence of complete DTR Ni accounting prevented verification of this conclusion. The final magnetics concentrates produced were typically 60-65% Fe and 2-3% Ni.

13.6.3.2 PFS Bench Testing

As part of the 2022 pilot plant, bench-scale testing was conducted to evaluate the impact of magnetic field strength at various regrind sizes on magnetic separation performance. Testing was completed in a Davis tube using the magnetic concentrate generated during the primary magnetic separation pilot plant. Metallurgical accounting was conducted on a total nickel basis. The lower magnetic strength Davis tube tests were compared to Davis tube results at 3500 G, which were used as a proxy for DTR recovery, as the regrind sizes selected were reasonably close to the DTR procedure grind size.

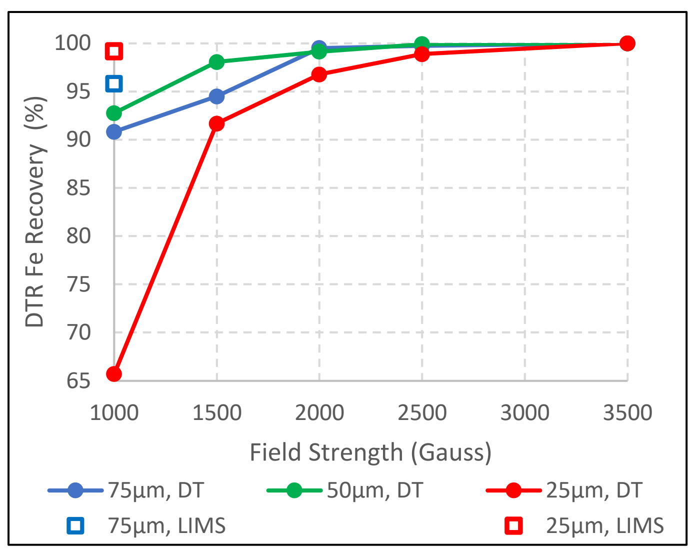
Figure 13-37 and Figure 13-38 present the DTR Ni and DTR Fe recoveries, respectively, and indicate the potential for loss of DTR Ni recovery at lower field strengths. Figure 13-39 and Figure 13-40 present the corresponding mass pulls and magnetics grades. All regrind sizes demonstrate increasing mass pulls up to 2,000 G. For the coarser regrind sizes, this increasing mass pull is accompanied by decreasing iron grade, indicating recovery of unliberated gangue minerals associated with magnetite. In contrast, the 25 µm regrind size does not show decreasing grade with increasing mass pull, indicating capture of well liberated magnetite particles. It is suspected that the finer particles have insufficient mass to overcome the hydraulic drag within the Davis tube. It is noted that the washing conditions utilized in a Davis tube are more aggressive than those utilized in commercial LIMS units. Thus, the potential for DTR recovery loss at lower field strength is likely overstated in these tests. This is evidenced by the results from bench-scale LIMS (2-stage cleaner and single cleaner-scavenger) which are overlaid on the same figures as the Davis tube tests.

Figure 13-37: Impact of Magnetic Field Strength and Regrind Size on DTR Ni Recovery



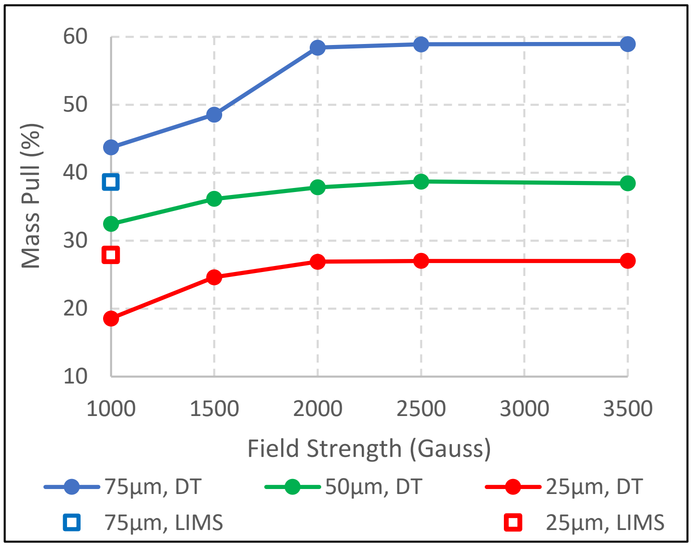
Source: FPX, 2023.

Figure 13-38: Impact of Magnetic Field Strength and Regrind Size on DTR Fe Recovery



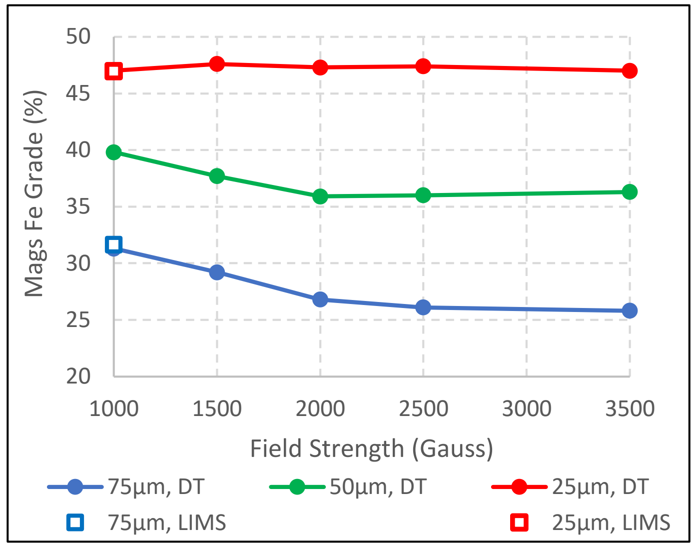
Source: FPX, 2023.

Figure 13-39: Impact of Magnetic Field Strength and Regrind Size on Mass Pull



Source: FPX, 2023.

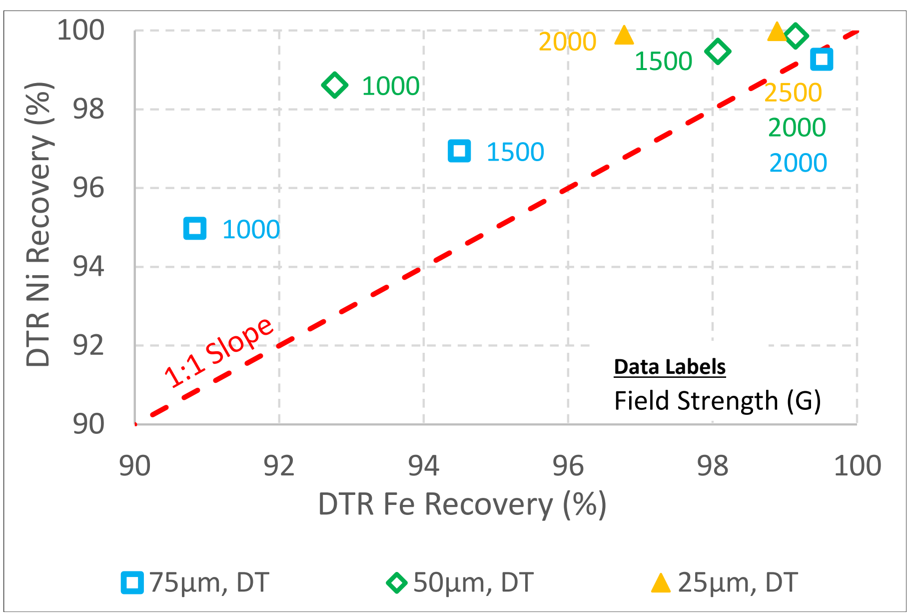
Figure 13-40: Impact of Magnetic Field Strength and Regrind Size on Magnetic Concentrate Iron Grade



Source: FPX, 2023.

Figure 13-41 presents the DTR Ni and DTR Fe recoveries from these tests. As observed in the primary grind (Figure 13-27), DTR Fe shows more recovery loss relative to DTR Ni at lower field strengths. Again, this is expected due to the lower magnetic susceptibility of magnetite compared to awaruite.

Figure 13-41: Impact of Magnetic Field Strength and Regrind Size on Mass Pull



Source: FPX, 2023.

13.6.3.3 PFS Pilot Plant Testing

13.6.3.3.1 2021 Pilot Plant

The 2021 pilot plant processed the primary magnetic separation concentrate generated during the same campaign at SGS Lakefield. The initial regrind target size was 25 µm and was followed by two sequential cleaner stages operating at 1,000 G. DTR Ni recoveries were excellent at 99.5%, but the resultant magnetics concentrate had an iron grade lower than was targeted for flotation feed (29% versus >50% Fe). The lower grade was primarily due to insufficient cleaning. Ancillary magnetic separation tests shows that an additional three LIMS cleaning stages increased the iron grade to 44-48% and that a Davis tube (and its associated thorough washing) was able to increase the iron grade to 52%. Although additional cleaning appeared to be able to approach the target flotation feed iron grade, the pilot plant proceeded with a light second regrind followed by a recleaner magnetic separation stage.

The second regrind resultant size was 18 µm and was followed by four sequential recleaner stages operating at 1,000 G. DTR Ni recovery was lower than anticipated at 95%. The DTR Fe recovery, at 90%, was notably lower than the DTR Ni recovery, suggesting an insufficient magnetic field strength (as discussed in Section 13.6.3.2). The magnetics concentrate produced had grades of 52% Fe and 2.1 Ni%. A total of 52 kg of magnetic concentrate was produced from the 2021 LOM composite (main sample) and was the feedstock used in the subsequent bench-scale flotation test program.

Both the first and second regrind were conducted in horizontal stirred media mills operating in open circuit; preferential grinding during these stages was not evaluated.

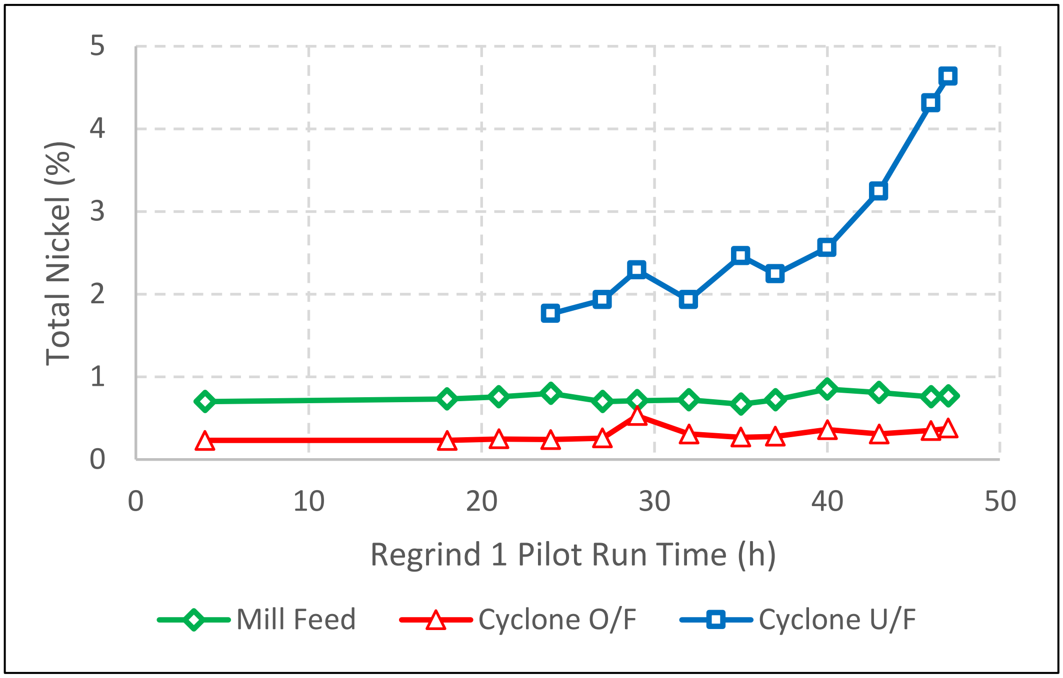
13.6.3.3.2 2022 Pilot Plant

The 2022 pilot plant processed the primary magnetic separation concentrate generated during the same campaign at Corem. As with the 2021 pilot plant, an initial regrind size of 25 µm was selected. Magnetic separation consisted of two sequential cleaner stages and a cleaner-scavenger stage all operating at 1,000 G. DTR Ni recoveries were good at 98.8%. The resultant magnetics concentrate had an iron grade of 46%. Ancillary Davis tube tests on the magnetics yield negligible upgrading indicating good cleaning in the pilot plant. To further upgrade the iron, the pilot cleaner magnetic concentrate was subjected to a light second regrind followed by a recleaner magnetic separation stage.

The second regrind resultant size was 22 µm, and was followed by three sequential recleaner stages operating at 1,200 G and a recleaner scavenger stage operating at 2,000 G. Although the difference between the first regrind size (25 µm) and the second regrind size (22 µm) was minimal, it did represent additional grinding work as the cleaner magnetics was biased coarser (~40 µm) than the cleaner feed (first regrind discharge) due to magnetic separation preferentially recovering coarser particles. DTR Ni recovery in the recleaner circuit was excellent at >99.9%, demonstrating the effectiveness of the slightly higher field strength in the recleaner scavenger which contributed an additional 0.7% DTR Ni recovery. The magnetic concentrate iron grade was 52%. The concentrate grade profile in the first/second/third recleaner was 50.5%/51.8%/52.3%, respectively, indicating minimum upgrading up the third stage. A Davis tube test on the third stage magnetics did not result in any additional cleaning.

The first regrind pilot plant was run in a ball mill operating in closed circuit with a hydrocyclone. Unlike the primary grind, the first regrind was run at a grind size that permitted the use of a pilot-scale hydrocyclone that operated at cyclone feed pressures more representative of industrial operation. A significant circulating load of nickel was observed which increased steadily over the duration of the pilot plant and never reached steady-state, as can be seen in Figure 13-42. Following completion of the pilot run, the mill contents were recovered and subjected to further investigations. Optical microscopy of the mill cleanout contents highlighted the presence of relatively large diameter (>100 µm), flat flakes of awaruite indicating that rather than breaking, awaruite deformed in the milling circuit. Awaruite's malleability is viewed as an opportunity as it is another property that differentiates awaruite from the gangue minerals and can potentially be exploited to improve mineral processing (discussed further in Section 13.7). The differential breakage properties of awaruite and gangue minerals is suspected to lead to preferential breakage along grain boundaries, leading to improved awaruite liberation.

Figure 13-42: Total Nickel Grades Over the Course of the First Regrind Pilot Plant



Source: FPX, 2023.

As generating sufficient awaruite concentrate was one of the primary objectives of the pilot plant, the awaruite in the first regrind mill cleanout products was targeted for recovery. Approximately 120 kg of milling cleanout products were batch ground to a P_{80} of 45 µm, processed by magnetic separation, and then the magnetics concentrate was processed by flotation to produce an awaruite concentrate. Although these unit operations were focused solely on generating concentrate, rather than collecting metallurgical data, it is notable that combined DTR Ni recovery in magnetic separation and flotation was estimated at 94% and produced a 66% Ni concentrate, indicating no challenges in processing this material through the proposed Baptiste flowsheet provided the circulating load of nickel in milling is properly managed.

As the quantity of awaruite concentrate generated through reprocessing of the first regrind mill cleanout was sufficient to meet the needs of the downstream hydrometallurgical refinery testing program, no further pilot-scale processing of the second regrind magnetic concentrate was necessary.

13.6.3.4 Variability Testing

The regrind variability program was part of the overall magnetic separation variability program, described in Section 13.6.2.4.1, and processed the magnetic concentrate generated during the primary magnetic separation stage.

The variability program used a standardized testing protocol and included batch grinding to first regrind P_{80} of 65 µm, following by a cleaner-cleaner-scavenger magnetic separation, with both separators run at 1,500 G. The cleaner magnetic concentrate was then subjected to a second regrind to ~18 µm, followed by a three-stage recleaner and single stage cleaner-scavenger magnetic separation, all at 1,500 G. The grind size and magnetic separation conditions are reflective of those selected for the PFS process plant design.

The two-stage regrind and cleaner/recleaner magnetic separation was selected following an engineering review of the PEA design which envisioned a single regrind stage to reduce from 300 to 25 µm. The required reduction ratio in the PEA is beyond what can be efficiently processed using stirred media mills, which is recommended for the fine regrind stage. A two-stage regrind allowed the most efficient milling technology to be selected for the required grinding duty and also allowed rejection of additional gangue at an intermediate regrind size, thereby reducing the load on the final regrind.

The results from both the cleaner and recleaner stages were consistent across all eight samples tested. The combined DTR Ni recoveries from both stages range from 98.1 to 99.7%, with a median of 99.0% (99.7% cleaner stage and 99.3% recleaner stage). DTR Fe recoveries were similar. The iron grade of the magnetic concentrates was similarly consistent, ranging from 29 to 34% in the cleaner and 50-55% in the recleaner. In aggregate, the variability results were consistent with the 2022 pilot plant results and highlight the lack of variability within the deposit when accounting on a DTR Ni and DTR Fe basis. The grades of the recleaner magnetic concentrates are provided in Table 13-7.

Table 13-7: Recleaner Magnetic Concentrates Produced in Variability Testing

Sample	Ni (%)	DTR Ni (%)	Fe (%)	DTR Fe (%)	MgO (%)
Comp B	2.17	2.17	55.5	55.3	8.5
Comp C	2.41	2.40	53.1	52.9	10.1
Comp D	2.94	2.94	51.2	51.1	10.6
Comp E	2.38	2.37	52.7	52.4	9.1
Comp F	2.00	2.00	52.2	52.1	10.9
Comp X	2.31	2.29	52.7	52.4	10.9
2021 LOM Comp	2.43	2.42	53.2	53.0	10.8

13.7 Gravity Separation

The original metallurgical test programs from 2010 to 2012 considered gravity separation for the final upgrading of the magnetic concentrate; that flowsheet option is no longer being pursued and is not discussed in this report. This section discusses gravity separation in the context of recovering DTR Ni from the circulating load of milling circuits operated in closed circuit with a hydrocyclone. Once recovered from the mill circulating load, any gravity concentrate would report downstream to the existing magnetic separation and/or flotation operations for final upgrading.

As discussed in Section 13.6.3.3.2, a significant circulating load of nickel was observed in the first regrind milling circuit of the 2022 pilot plant. To investigate if this circulating load could be readily recovered through gravity separation, a sample of the cyclone underflow from the pilot plant was subjected to centrifugal concentration using a bench-scale Knelson concentrator. A single pass through the concentrator recovered 30% of the contained nickel into a gravity concentrate grading 46% Ni with a 2% mass pull. An additional four passes cumulatively recovered an additional 45% of the nickel into a gravity concentrate grading 21% Ni with an 8% mass pull. These results clearly demonstrate the ability to use gravity to concentrate and remove nickel contained in the mill circulating load.

In the 2022 pilot plant primary grinding circuit, the mill discharge DTR Ni grade ranged between 1.6-2.4 times greater than that of the mill feed. In an industrial operation with higher hydrocyclone pressures, this DTR Ni grade would be expected to increase. A size-by-size assay of the mill discharge revealed that an estimated 78% of the DTR Ni contained was present in the 300 µm fraction. This indicates that DTR Ni could also be readily removed from the circulating load through screening a portion of the circulating load and bleeding the undersize out toward magnetic separation.

A further characterization of the gravity recoverable nickel was determined by running the outcrop material through a modified gravity recoverable gold (GRG) test. The GRG is an industry standardized test to assess the deportment of gravity gold at various milling sizes using a centrifugal concentrator. The procedure was modified to match the primary grind and regrind sizes considered for Baptiste. Results indicate that approximately 25% of the DTR Ni was recovered at the primary grind size (250 µm), an additional 56% at the first regrind size (65 µm), and further 6% at the second regrind size (18 µm).

Cumulatively, this work suggests that screening and/or a centrifugal concentrator are appropriate techniques to reduce or prevent awaruite accumulation in the mill circulating load. Operating grinding mills in an open circuit configuration or closed with screens are other viable options.

13.8 Flotation

13.8.1 PEA Bench Testing

As part of the 2018 metallurgical testing that supported the 2020 PEA, a simple flotation regime was developed and thoroughly tested to prove that flotation is a viable route to recover awaruite from a concentrate generated through magnetic separation. The flotation regime utilized a mildly acidic conditioning stage to activate the awaruite surface, followed by addition of a conventional xanthate collector to render the awaruite recoverable. The PEA testing successfully achieved 90-93% nickel recoveries from the flotation feed and produced a high-grade concentrate grading over 60% Ni. The PEA testwork identified multiple optimization opportunities, primarily aimed at improving overall recovery and optimizing reagent usages. The PEA testing was limited to flotation feed material sourced from the PEA starter pit composite.

13.8.2 PFS Bench Testing

13.8.2.1 Feed Characterization

The primary objective of the PFS flotation test program was to further optimize the flotation regime developed during the PEA test program and to demonstrate successful flotation performance on a metallurgical composite representative of the LOM plant feed. The 2021 SGS Lakefield pilot plant processed the 2021 LOM composite (main sample) through primary and cleaner magnetic separation to produce 52 kg of magnetite concentrate which then served as the flotation feed for the 2022 flotation bench testing program executed by SGS Burnaby. The PFS bench-scale testing consisted of 55 open circuit rougher and cleaner flotation tests, as well as three locked cycle rougher and cleaner flotation tests.

The composition of the magnetic concentrate/flotation feed is presented in Table 13-8. The flotation feed, as inferred from whole ore mineralogy and flotation tails mineralogy, is predominantly magnetite and serpentine minerals. The serpentine minerals are associated with magnetite as most of the liberated serpentine minerals are rejected upstream through magnetic separation. The majority of nickel is present as awaruite. Chromium exists as an unidentified chromium-iron spinel mineral. As discussed in Section 13.6.2.3.1, the loss of coarse awaruite during primary milling in the magnetic separation plant is suspected to have contributed to lower than expected flotation feed nickel grades. This was confirmed in the variability testing where the recleaner magnetic concentrate for the 2021 LOM composite sample graded 2.43% Ni and 53.0% Fe versus 2.11% Ni and 52.3% Fe in the 2021 SGS Lakefield pilot plant magnetic concentrate used in the flotation test program.

Table 13-8: Composition of PFS Bench Testing Flotation Feed

Element (%)	Fe	Ni	Cr_2O_3	SiO_2	MgO	Al_2O_3	CaO	MnO
Flotation Feed	52.3	2.11	3.30	8.61	9.25	0.40	0.17	0.24

13.8.2.2 Open Circuit Testing

Over the course of the flotation test program multiple variables were evaluated to improve flotation performance and reagent usage. The flotation conditions for a later stage test, accounting for optimizations made during testing, are presented in Table 13-9. Notable changes made to the PFS flotation regime, relative to the PEA flotation regime, are:

- Removal of copper sulphate as an activator, which was not demonstrated to improve recovery.
- Increase of target pH during acid conditioning stage and rougher-cleaner stages to reduce acid consumption, reduce xanthate degradation rates, and provide less corrosive conditions for materials of construction considerations.
- Addition of a light polishing regrind step intended to scratch any passivating layer that may have build-up on the awaruite mineral during the storage of the magnetic concentrate.

Table 13-9: Typical Flotation Conditions Used In PFS Bench Testing (Test F35)

Stage	Reagents added (g/t)				Time (min)	
	Control pH	H_2SO_4	SIPX	Dow 250	Condition	Froth
Ceramic Grind	-	-	-	-	2	-
Acid Conditioning	4.0-4.2	9,217	-	-	10	-
Collector Conditioning	4.8-5.0	5,760	150	-	5	-
Rougher 1	4.8-5.0	323	-	550	1	2
Rougher 2	4.8-5.0	230	60	-	1	2
Rougher 3	4.8-5.0	323	60	-	1	2
Rougher 4	4.8-5.0	323	40	-	1	4
Rougher 5	4.8-5.0	571	40	-	1	4
Cleaner 1	4.8-5.0	91	15	-	1	3
Cleaner 2	4.8-5.0	43	15	100	1	3
Cleaner 3	4.8-5.0	59	15	-	1	4
Total	**-**	**16,940**	**395**	**650**	**23**	**24**

The resultant metallurgical balance, corresponding to the flotation regime presented in Table 13-8, is presented in Table 13-9 and shows a nickel recovery of 84.1%, a rougher tailings grade of 0.25% Ni, and a final concentrate grade of 57.3% Ni. These results are comparable to those observed in the PEA bench testing program with typical results of a 90.1% nickel recovery, a rougher tailings grade of 0.25% Ni, and a final concentrate grade of 63.4% Ni. The reduction in recovery for the PFS program is primarily attributed to a lower head nickel grade as both programs and feed samples produced comparable rougher tailings grades.

Table 13-10: Typical Open Circuit Flotation Test Results (Test F35)

Product	Weight		Assays (%)		Distribution (%)	
	g	%	Ni	Fe	Ni	Fe
Overall Metallurgical Balance						
Cleaner Concentrate	15.6	3.15	57.3	26.3	84.1	1.6
Cleaner 3 Tail	1.4	0.28	2.38	46.5	0.3	0.3
Cleaner 2 Tail	9.8	1.98	1.27	45.1	1.2	1.7
Cleaner 1 Tail	56.4	11.4	0.91	49.1	4.8	10.8
Rougher Tail	411	83.2	0.25	53.1	9.6	85.6
Head (calculated)	495	100	2.15	51.6	100	100
Cumulative Metallurgical Balance						
Cleaner 3 Concentrate	15.6	3.2	57.3	26.3	84.1	1.6
Cleaner 2 Concentrate	17.0	3.4	52.8	28.0	84.4	1.9
Cleaner 1 Concentrate	26.8	5.4	34.0	34.2	85.6	3.6
Rougher Concentrate	83.2	16.8	11.6	44.3	90.4	14.4
Rougher Tail	411	83.2	0.25	53.1	9.6	85.6

13.8.2.3 Locked Cycle Testing

Table 13-10 presents the results from locked cycle testing using conditions comparable to those presented for open circuit testing in Table 13-9. The projected metallurgy from the locked circuit testing is also presented and reports a final concentrate nickel recovery of 85.3%, a rougher tailings grade of 0.28% Ni, and a final concentrate grade of 59.0% Ni. The results are comparable to the open circuit test results presented in Table 13-9 with only a small increase in the rougher tailings grade, indicating that the majority of nickel contained in the recycled cleaner tailings was recovered while maintaining final concentrate grades.

The locked cycle testing also reports the percentage of nickel which is dissolved during the acidic conditioning and flotation stages. While dissolved nickel is not reported in the open circuit metallurgical balance, other open circuit testing results show similar extents of nickel dissolution as was found in the closed circuit testing. The dissolution of a small amount of nickel during the acid conditioning stages is expected as slow dissolution of awaruite has been observed in other Baptiste metallurgical programs which specifically targeted awaruite dissolution.

Table 13-11: Locked Cycle Flotation Test Results

Product	Weight		Assays (%)		Distribution (%)	
	g	%	Ni	Fe	Ni	Fe
Overall Metallurgical Balance						
Cycle 1: Cleaner 4 Concentrate	13.4	0.53	65.5	24.16	15.7	0.25
Cycle 2: Cleaner 4 Concentrate	13.7	0.55	62.5	24.15	15.4	0.26
Cycle 3: Cleaner 4 Concentrate	17.6	0.70	57.8	25.36	18.2	0.35
Cycle 4: Cleaner 4 Concentrate	15.8	0.63	60.5	25.46	17.1	0.31
Cycle 5: Cleaner 4 Concentrate	15.5	0.62	54.2	25.02	15.1	0.30
Cycle 5: Cleaner 4 Tail	8.1	0.32	8.64	28.3	1.26	0.18
Cycle 5: Cleaner 3 Tail	11.5	0.46	2.10	35.9	0.43	0.32
Cycle 5: Cleaner 2 Tail	21.2	0.85	1.26	38.1	0.48	0.63
Cycle 5: Cleaner 1 Tail	56.3	2.2	0.61	42.5	0.61	1.85
Cycle 1: Rougher Tail	434.2	17.3	0.25	53.1	2.0	17.9
Cycle 2: Rougher Tail	459.9	18.3	0.32	52.6	2.6	18.8
Cycle 3: Rougher Tail	487.0	19.4	0.29	54.1	2.5	20.4
Cycle 4: Rougher Tail	460.0	18.3	0.28	52.4	2.3	18.7
Cycle 5: Rougher Tail	493.6	19.7	0.35	51.6	3.1	19.7
Cycle 1-5: Flotation Solution	-	-	-	-	3.3	0.0
Projected metallurgy (based on cycles 3/4)						
Cleaner 4 Concentrate	33.4	3.4	59.0	25.4	85.3	1.7
Rougher Tail	947.0	96.6	0.28	53.3	11.6	98.3
Flotation Solution	-	-	-	-	3.1	0.0
Rougher Feed	980.4	100.0	2.36	52.4	100	100

13.8.2.4 Recovery of Dissolved Nickel

Nickel dissolved in flotation conditioning was identified as a target for downstream recovery, which is discussed further in Section 13.9. A necessary pre-requisite for the economic recovery of dissolved nickel is an acceptably high nickel concentration in solution. Increasing the nickel concentration in solution requires a) maintaining a tight water balance in flotation and b) achieving acceptable flotation performance when conducted in a solution with elevated concentrations of dissolved solids. The additional testwork to support these two items are discussed in this section.

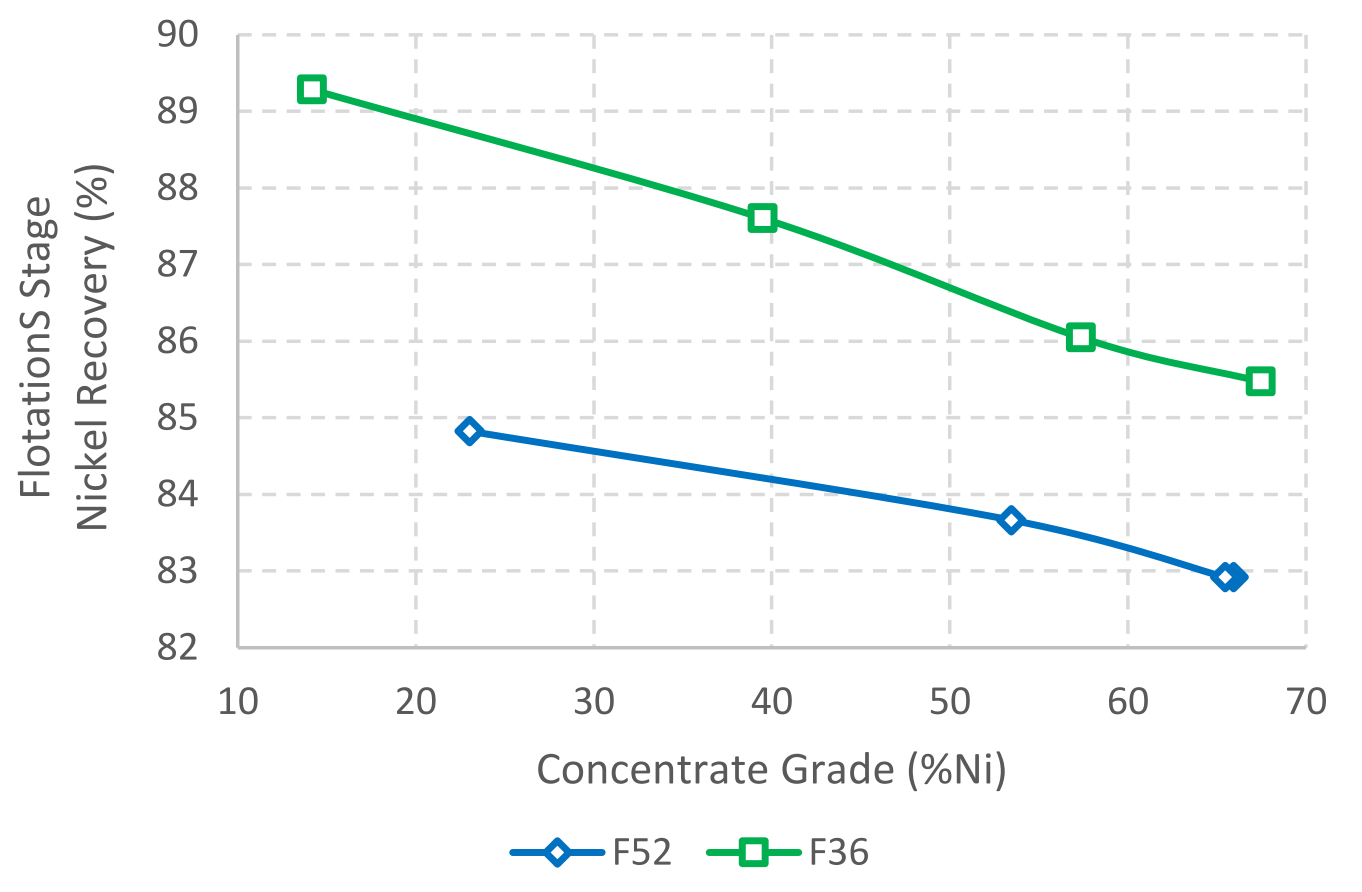
Achieving a tight water balance in the flotation circuit requires the flotation feed, tailings, and concentrate to have acceptable solid-liquid separation properties, which will allow limiting the new water entering the circuit, capturing the flotation solution and then recycling the solution back to flotation conditioning for dilution. To assess the solid-liquid separation properties, bench-scale thickening, vacuum filtration, and vacuum filtration testing were conducted on both the flotation tailings and the flotation concentrate. Both materials showed acceptably low moisture content with ~30%w/w moisture for the thickened flotation tailings, ~15%w/w for the pressure-filtered flotation tails, and lower values for the flotation concentrate. Further, the settling and filtration rates observed for both materials were relatively high, indicating modest dewatering equipment requirements. The solid-liquid separation properties of the flotation feed can be reasonably expected to be comparable to the flotation tailings. These combined results indicate that achieving a tight water balance around flotation is feasible, thus permitting an increase in the nickel concentrations in solution.

As nickel concentrations in solution increase, so do other acid-consuming elements that will dissolve in flotation conditioning. Magnesium from serpentine minerals accounts for 80-90% of acid consumption in flotation and leads to the formation of magnesium sulphate as the dominant dissolved species. To assess the impact of elevated magnesium sulphate concentrations on flotation performance, additional open circuit testing was conducted in a magnesium sulphate brine, as opposed to water. Brine concentrations of 10 to 20% $MgSO_4$ were tested with no signs of degraded flotation performance relative to the open and locked circuit testing conducted in tap water. The only notable difference between the brine tests and the water tests were a decrease in required frother dosing rates in the brine tests. This is expected given the self frothing ability of high total dissolved solids (TDS) solutions. In addition to higher flotation solution TDS, operating with a tight water balance is expected to increase the flotation operating temperature slightly. Additional open circuit tests operating at temperatures up to 50°C were tested with no degradation in flotation performance observed. The higher temperature operation demonstrated a significant increase in rougher grades (from ~10% to ~30% Ni) which suggests a potential for a reduction in the flotation cleaning requirements.

13.8.2.5 Concentrate Grade vs Recovery Relationship

Final concentrate grades produced in the optimized open circuit and stabilized locked cycled flotation testing produced final concentrate grades ranging between 55-68% Ni and 59-61% Ni, respectively. To assess the impact of concentrate grade on recovery, grade-recovery curves from open circuit tests were reviewed. Figure 13-46 presents the grade-recovery curve of two representative tests and shows a relatively flat grade-recovery curve, with an estimated increase in recovery of 0.3% for a 5% decrease in concentrate grade. Grades of the final concentrate are discussed further in Section 13.10.1.

Figure 13-43: Grade-Recovery Curve from Open-Circuit Testing



Source: FPX, 2023.

13.8.2.6 Flotation Tailings Characterization

Additional characterization of select flotation tailings was conducted to understand the nature of nickel losses and to assess the flotation tailings as a potentially saleable iron ore concentrate.

Mineralogy on flotation tailings identified awaruite as the dominant mineral species present, accounting for approximately 95% of the contained nickel. Of this, greater than 30% of the awaruite was present in liberated or sub-liberated particles and were generally in the 2-10 µm size range. This is not unexpected given particles in this size class are challenging to recover in flotation, particularly when they are not spherical, and therefore have a higher surface area to mass ratio. The balance of awaruite was present as fine inclusions in both magnetite (20% of total awaruite and generally <2 µm) and serpentine (30% of total awaruite and generally <5 µm). Notably, coarser awaruite grains were not observed despite a mineralogical study of the flotation feed indicating a measured awaruite P_{50} value of 38 µm. This indicates very efficient recovery of coarse, liberated awaruite particles in the flotation process.

The composition of typical flotation tailings, produced from both the PFS and PEA flotation programs, is presented in Table 13-12. The 2010 LOM composite shows an iron grade of 54.4%, which is notably lower than that produced from the PEA starter pit composite, which likely reflects the finer magnetite grain size distribution in the feed samples of each material (as shown previously in Figure 13-15). Mineralogy of the flotation tailings indicated that approximately two-thirds of the magnetite was liberated. Preliminary Davis tube separations at varying field strengths indicated that at weak magnetic field strengths 83% of the iron could be recovered to a concentrate grading 57% Fe. Based on this mineralogy, further iron upgrading is likely possible; however, only approximately one-third of the chromium minerals were liberated, indicating that producing a low chromium-iron concentrate may be challenging. Therefore, any iron concentrate produced is likely best suited to stainless steel production, where chromium is a value element, rather than carbon steel production, where it is an impurity element.

Table 13-12: Flotation Tailings Composition

Element (%)	Fe	Si	Mg	Cr	Ni	S	Fe_3O_4 (Satmagan)
2021 LOM Composite	54.4	4.18	4.94	2.11	0.28	0.02	68.6
PEA Starter Pit Composite	61.0	1.81	2.26	1.92	0.33	-	-

13.8.2.7 Alternative Flotation Regime

In addition to the flotation regime discussed in Section 13.8.2.2, a second flotation regime was tested in open circuit and locked circuit tests. The alternative flotation regime was similar to the primary regime but included additional activators to permit the use of an elevated operating pH for acid conditioning and flotation with commensurately reduced acid consumption. The locked cycle results from the alternative regime are presented and compared to the standard regime in Table 13-13. While the alternative flotation regime can achieve a 25% reduction in acid consumption, it is also accompanied by a slight reduction in recovery. While technically viable, the alternative flotation regime is not being pursued in the PFS regime as the acid consumption savings are insufficient to compensate for the reduced recovery. Further, any reduction in acid consumption in flotation would be expected to be offset by a commensurate increase in acid consumption in the downstream flotation leach section. Effectively, flotation conditioning serves as a pre-leach stage for the flotation tailings leach.

Table 13-13: Comparison of Standard and Alternative Flotation Regime Locked Cycle Flotation Results

Regime	Configuration	Acid Consumption (kg/t)	Rougher Tails Grade (%Ni)	Recovery to Final Cleaner Con. (%)	Final Cleaner Con. Grade (%)	Nickel Dissolution Extent (%)
Standard	Locked cycle	18.0	0.28	85.3	59.0	3.1
Alternative	Locked cycle	13.6	0.35	83.3	54.9	2.1

13.8.2.8 Recommendations

Review of the PFS flotation program has identified the following items which are recommended to be investigated further in the next phase of study:

- Characterisation of the flotation tailings indicated that approximately one-quarter of the nickel contained in the flotation tailings is fine liberated awaruite is potentially recoverable. This would represent an approximate additional 2% overall nickel recovery to the flotation concentrate. Additional flotation work targeting this fine awaruite is recommended. It is noted that any improvement in flotation recovery would be expected to be accompanied by a reduction in the flotation tailings leach recovery. This would allow a value-based flowsheet decision as to the best means to recover nickel in consideration of both process complexity, capital cost, and operating cost.

- Xanthate dosages in flotation are relatively high and partially reflect the accelerated xanthate degradation rates that occur under acidic pH conditions. Investigations of alteration flotation collectors, which are more stable in mildly acidic environments, is recommended.
- Flotation variability testing is recommended to support the LOM composite results completed in the PFS. Note that flotation testing on products of the magnetic separation variability campaign is ongoing, but are not complete at the time of writing this report.
- Limited testing aimed at further upgrading the iron grade of the flotation tailings has been completed to date. Additional testing, to produce a potentially saleable iron ore product, is recommended.
- 7-17% of the total nickel in the whole ore is contained as nickel sulphides which are not presently targeted for recovery. Flotation testing conducted on the magnetic separation tailings is recommended to determine if these nickel minerals can be economically recovered to a saleable concentrate.

13.9 Flotation Tailings Leach

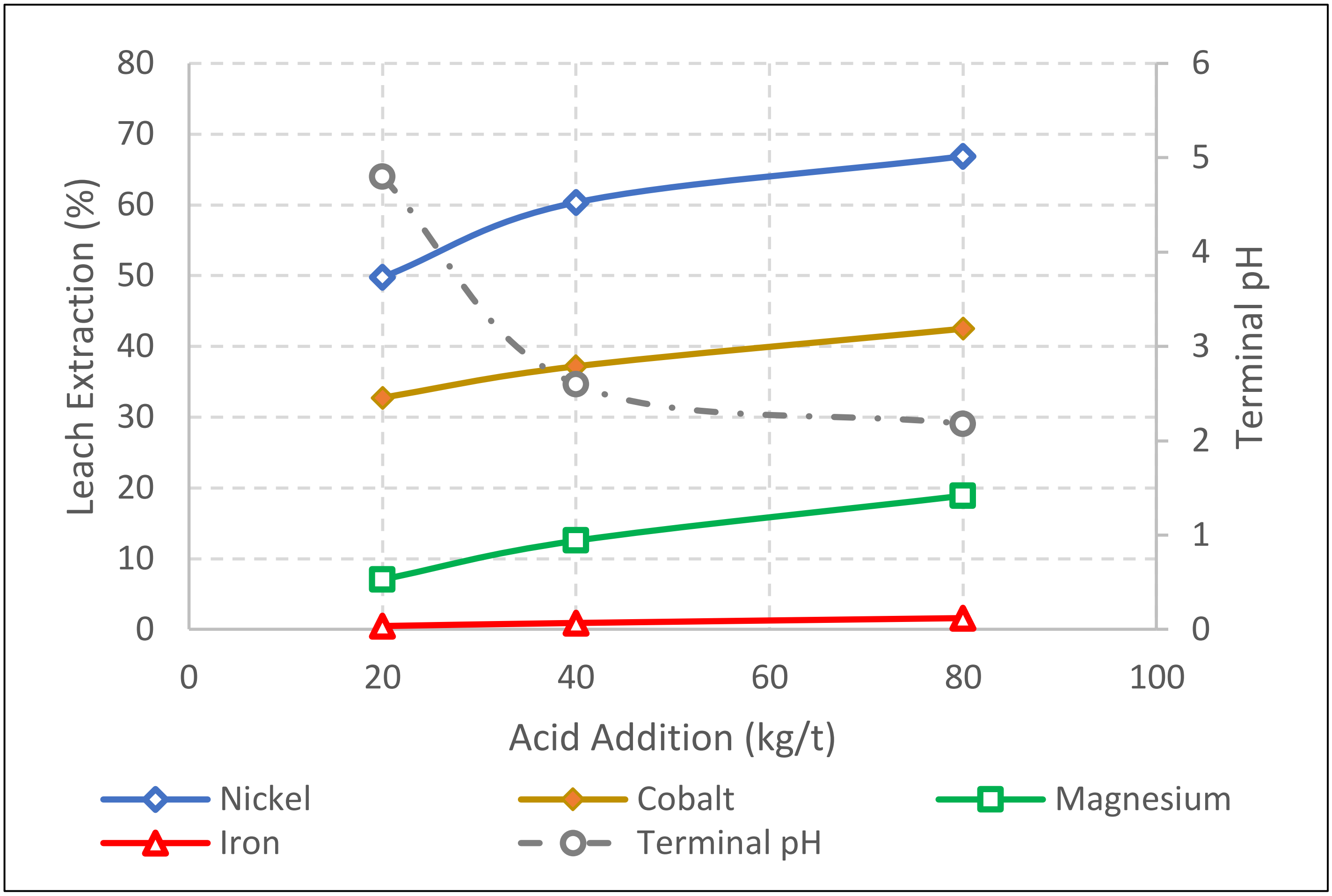
As the flotation tailings contained fine, liberated awaruite particles, investigatory testing was completed to assess the leaching potential of the flotation tailings. The dissolution of awaruite during flotation conditioning, with its mildly acidic conditions and short residence times, was indicative of the reactivity of awaruite. To assess the quantity of awaruite leachable in a selective leach, with minimal dissolution of iron and magnesium minerals, a selective assay for metallic and sulphide nickel was undertaken. The results from the soluble nickel assays are presented in Table 13-14 and show approximately two-thirds of the contained nickel is exposed and available for leaching.

Table 13-14: Selective Nickel Assay on Flotation Tailings

Assay	Value (%)
Total Nickel	0.28
Soluble Nickel	0.18
Insoluble Nickel (calculated)	0.10

Results from a series of leaching tests undertaken at varying sulphuric acid addition rates are summarized in Figure 13-44. The tests were conducted at atmospheric pressure and ambient temperature with an eight-hour residence time. The tests show 20 kg/t acid addition results in 50% nickel leach extent with minimal magnesium co-leaching, negligible iron co-leaching, and high terminal pH, indicating negligible free acidity in the leach solution. Higher leach recoveries of up to 70% were achieved with increased acid additions; however, an economic analysis showed acid addition rates above 20 kg/t provided limited additional benefit as the incremental nickel recovery did not outweigh the additional acid addition and downstream solution purification reagent costs. Additional testing to optimize acid addition rates and minimize magnesium co-leaching are in progress but were not complete at the time of writing this report.

Figure 13-44: Impact of Acid Addition on Metal Recovery in Flotation Tailings Leach



Source: FPX, 2023.

Testing to further process the leach solution, from both flotation conditioning and flotation tailings leaching, to a MHP have not been completed. However, the solution purification and nickel/cobalt precipitation steps required are conventional and well established industrially. The primary impurities in the leach solution are magnesium and iron, with negligible quantities of aluminum, manganese, copper, and zinc. Qualitative filtration testing conducted on the leach residue slurry demonstrated good filtration and washing characteristics, permitting the recovery of leach liquor from the leach residue. Given the industrial benchmarks for MHP production from leach solutions and given that MHP is a coproduct accounting for only 7% of nickel production (the balance being the flotation concentrate), the assumptions defining MHP production at this phase of study are considered reasonable. Additional testing is recommended for the next phase of study and should include leach optimization, leach residue dewatering and washing, solution purification, and MHP precipitation.

13.10 Product Quality

13.10.1 Awaruite Concentrate

Testing described in this report demonstrate that a high nickel grade awaruite concentrate can be produced by a mineral processing route consisting of sequential magnetic separation followed by flotation. Awaruite concentrates grading between 60-65% have been produced in flotation testing, with only a minor difference in recovery observed between these two concentrate grades. Concentrate specifications based on the PFS testing program for both these product grades, including impurity elements of concern for stainless steel production, are presented in Table 13-15. In addition to absolute elemental concentrations, elemental ratios of nickel-to-impurity are also presented as this metric is more meaningful when considering the impacts on stainless steel production (i.e., a higher nickel grade product would require a reduced amount of concentrate, which in turn would reduce the quantity of associated impurities added). The impacts of specific impurities are discussed in detail in Section 19.5 which concludes that awaruite concentrate should be able to substitute for standard FeNi in stainless steel production, and therefore attract similar payability to standard FeNi.

Discussions with a global stainless steel producer indicated the 65% Ni product would likely not attract a pricing premium relative to the 60% Ni product. For the purposes of the PFS, a concentrate grade of 60% Ni has been assumed. Although a 65% Ni concentrate grade is assessed to be technically feasible, the 60% Ni concentrate grade will provide more operational flexibility.

A briquetted product form is required for delivery into the stainless steel market. No briquetting testing has been completed for the PFS. Briquetting testing is recommended for the next phase of study to ensure a suitably resilient briquette can be produced and to confirm the assumed briquetting conditions.

Table 13-15: Awaruite Concentrate Product Specifications

Element	Unit	60% Ni Product	65% Ni Product
Nickel, Total (Ni)	Percentage	60	65
Nickel, Metallic (Ni)	Percentage	~60	~65
Iron, Total (Fe)	Percentage	24.6	24.2
Iron, Metallic (Fe)	Percentage	18.1	19.3
Sulphur (S)	Percentage	0.98	0.92
Carbon (C)	Percentage	0.10	0.09
Phosphorus (P)	Percentage	<0.01	<0.01
Silicon (Si)	Percentage	2.2	1.3
Chromium (Cr)	Percentage	0.06	0.04
Cobalt (Co)	Percentage	1.08	1.12
Copper (Cu)	Percentage	0.49	0.53
Magnesium (Mg)	Percentage	1.7	1.0
Nickel:Sulphur Ratio (Ni:S)	Ratio	61	71
Nickel:Cobalt Ratio (Ni:Co)	Ratio	56	58
Nickel:Copper Ratio (Ni:Cu)	Ratio	122	124

13.10.2 Mixed Hydroxide Precipitate

A MHP product was not produced in the PFS test program, however, testwork to produce MHP is recommended for the next phase of study. A predicted MHP product quality has been estimated based on comparing the solution chemistry observed during flotation tailings leach testing and comparing it the solution chemistry of benchmarked leach solutions along with their corresponding MHP grades, to other production routes used to generate MHP.

The nickel grade of the Baptiste MHP is expected to exceed 45% nickel and 0.7% cobalt and contain industry standard levels of sulphur and magnesium contamination. The Baptiste MHP is expected to be higher grade than MHP produced from nickel laterite feedstocks via the high-pressure acid leach (HPAL) process route, which typically range from 33-40% nickel. This is primarily due to the near absence of manganese minerals in the Baptiste Deposit, which are a material contributor to the impurity content in typical MHP derived from laterite feedstocks, thereby diluting nickel grades.

13.11 Recovery Model

Estimates of the overall circuit recovery were made and applied to the PFS economic analysis based on the cumulative PFS test work summarized in this report. Key components of the recovery model are summarized below:

- Primary Magnetic Separation: 95.0% stage DTR Ni recovery based on a median recovery of 93.5% in the variability bench testing program (Section 13.6.2.4.1) and an incremental recovery benefit of 1.5% due to preferential grinding in primary grinding (Section 13.6.2.3.2).
- Cleaner Magnetic Separation: 99.3% stage DTR Ni recovery based on a median recovery of 99.3% in the variability bench testing program (Section 13.6.3.4).
- Recleaner Magnetic Separation: 99.7% stage DTR Ni recovery based on a median recovery of 99.7% in the variability bench testing program (Section 13.6.3.4).
- Flotation – Recovery to Concentrate: 87.4% stage DTR Ni recovery based on a flotation tailings grade of 0.28% Ni as determined from the PFS flotation program locked cycle testing (Section 13.8.2.3). The constant tailings grade approach was selected to correct for the material quantities of coarse awaruite that were assessed to be lost during the 2021 pilot plant primary grind (as discussed in Section 13.6.2.3.1).
- Flotation – Nickel dissolution to solution: 3.1% dissolution extent based on flotation feed as determined from the PFS flotation program locked cycle testing (Section 13.8.2.3).
- Flotation Tailing Leach: 50% dissolution extent based on PFS tailings leach program (Section 13.9) and based on an acid addition rate of 20 kg/t flotation tailings.
- Dissolved Nickel: 88% recovery of dissolved nickel to the MHP product based on the mass balance of the solution recovery circuit, accounting for nickel losses in leach tailings residue washing, iron removal co-precipitation, and MHP precipitation loss to discharge solution. Nickel recovery parameters in each unit operation were based on industrial benchmarking of similar unit operations, most notably from existing high-pressure acid leach (HPAL) circuits to produce MHP from laterite feedstocks.

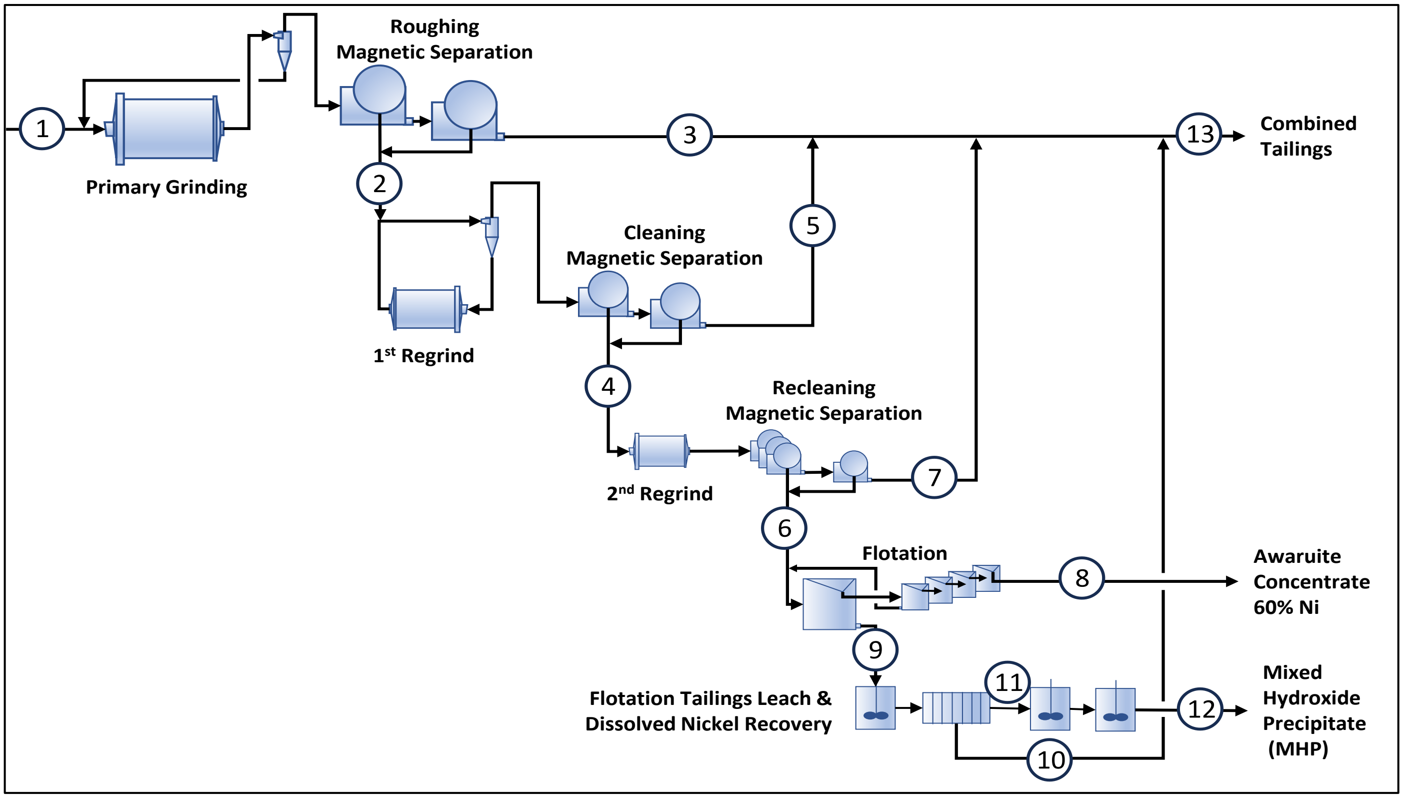
An example of the recovery model and circuit mass balance, using LOM ore grades, is presented in Table 13-16, along with the corresponding process flow diagram presented in Figure 13-45. Year-by-year recovery projections vary minimally, as presented in Figure 13-46. A second recovery projection is presented in Figure 13-47 showing recovery on a total nickel basis. Total nickel recovery is calculated from the DTR Ni recovery and the ratio of DTR Ni to total nickel in the feed ore.

Table 13-16: Life-of-Mine Ore Grade Mass Balance

Stage	ID #	Product	Stage Recovery (%)			Grade (%)		Overall Distribution (%)		
			Mass	DTR Ni	DTR Fe	DTR Ni	DTR Fe	Mass	DTR Ni	DTR Fe
Mill Feed	1	Whole Ore	-	-	-	0.13	2.39	100.0	100.0	100.0
Rougher Mag. Sep.	2	Concentrate	16.0	95.0	95.0	0.773	14.2	16.0	95.0	95.0
Rougher Mag. Sep.	3	Tailings	84.0	5.0	5.0	0.008	0.14	84.0	5.0	5.0
Cleaner Mag. Sep.	4	Concentrate	47.0	99.3	99.3	1.63	30.0	7.5	94.3	94.3
Cleaner Mag. Sep.	5	Tailings	53.0	0.7	0.7	0.010	0.19	8.5	0.7	0.7
Recleaner Mag. Sep.	6	Concentrate	57.5	99.7	99.7	2.83	52.0	4.3	94.1	94.1
Recleaner Mag. Sep.	7	Tailings	42.5	0.3	0.3	0.012	0.21	3.2	0.3	0.3
Flotation	8	Concentrate	4.1	87.4	2.0	60.0	24.6	0.18	82.2	1.8
Flotation	9	Tailings	95.1	9.5	98.0	0.28	53.2	4.1	8.9	92.2
Flotation	-	Dissolved Nickel	0.8	3.1	0.1	-	-	0.0	2.9	0.0
Flotation Tailings Leach	-	Leach PLS	1.1	50.0	0.0	-	-	0.0	4.5	0.0
Flotation Tailings Leach	10	Leach Residue	98.9	50.0	100.0	0.14	53.8	4.1	4.5	92.2
Dissolved Nickel Recovery[1]	11	Combined Feed	-	100	-	-	-	-	7.4	-
Dissolved Nickel Recovery[1]	-	Solution Loss	-	12	-	-	-	-	0.9	-
Dissolved Nickel Recovery[1]	12	MHP Product	-	88	-	-	-	-	6.5	-
Combined Tailings	13	Tailings	-	-	-	0.014	2.35	99.8	10.4	98.2

Note: Dissolved nickel and MHP are not magnetically recoverable, but for simplicity has been accounted for as DTR Ni as the nickel units originated from DTR Ni.

Figure 13-45: Mass Balance Flow Diagram



Source: FPX, 2023.

Figure 13-46: Yearly DTR Ni Recovery Projection

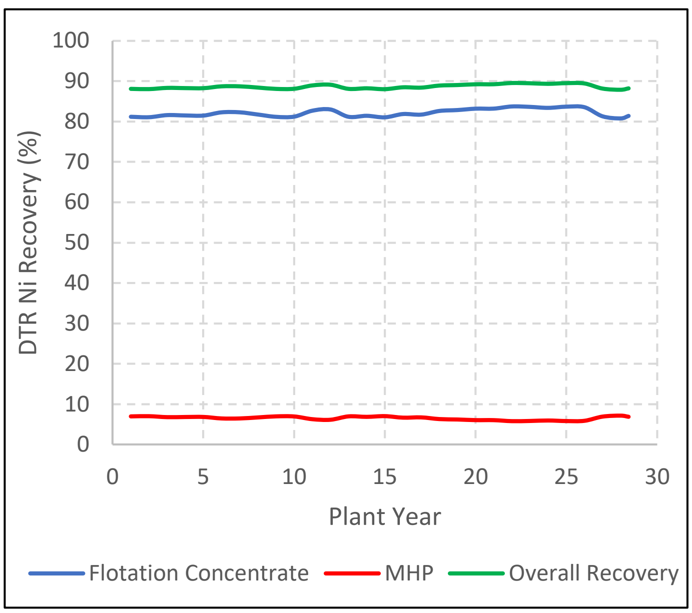
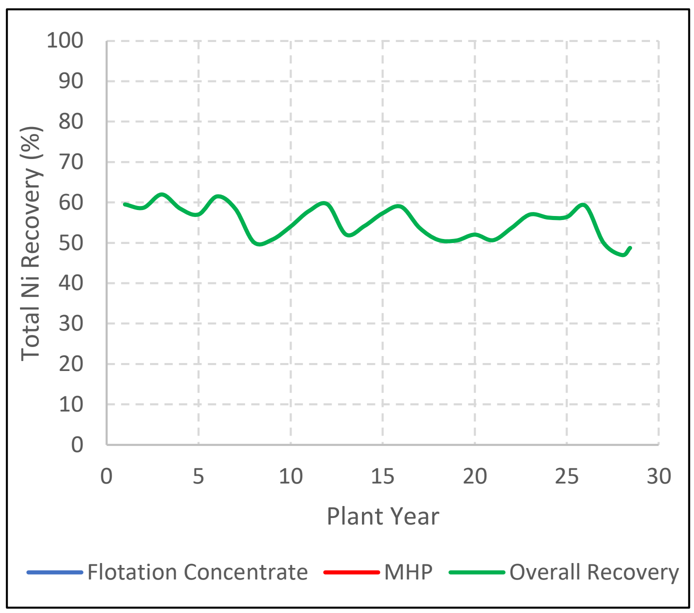


Figure 13-47: Yearly Total Ni Recovery Projection



Source: FPX, 2023.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

This section describes the methods used to produce an updated Resource Estimate for the Baptiste Deposit. No estimates of mineral resources or mineral reserves have been made for the Van, Sid and B targets.

14.2 Key Assumptions and Basis of Estimate

The sample database supplied for the Baptiste Deposit contains results from 99 surface drillholes completed since 2010 (Table 14-1), with 93% of all metres drilled having assay results, as shown in Table 10-1. In comparison to the 2020 Resource Estimate, the 2022 Resource Estimate incorporated an additional 17 diamond drillholes totalling 2,856 m from the 2021 drilling campaign. The average drillhole spacing in the Baptiste Deposit is 150 m.

Table 14-1: Summary of Drillhole and Sample Totals Used in 2022 Resource Estimate

Year	Holes	Metres	Samples	Assayed (m)	Comments
2010	7	1,710.8	638	1,533.5	Samples include 2012 resampling program
2011	35	10,863.6	5,199	10,176.5	Samples include 2012 resampling program
2012	32	16,346.8	4,153	15,410.9	-
2017	8	1,917.5	460	1,565.1	-
2021	17	2,855.9	723	2,536.7	-
Total	**99**	**33,694.6**	**11,173**	**31,222.7**	**-**

The 2022 resource model comprises a large, delta shaped volume that measures approximately 3 km in length, 150 to 1,080 m in width and extends to a depth of 540 m below the surface. The Baptiste Deposit remains open at depth over the entire system and is covered by an average of 12 m of overburden.

Davis Tube magnetically recoverable (DTR) nickel is the nickel content recovered by magnetic separation using a Davis Tube, followed by fusion with lithium borate and an XRF finish to determine the nickel content of the magnetic fraction; in effect, a mini-scale metallurgical test. The Davis Tube method is the industry standard for the quantitative analysis of magnetic minerals (e.g., SGS, 2009).

14.3 Geological Model

The 2022 updated geological interpretation was provided by Equity Exploration as a model of lithology (Figure 14-1). The model was created using Leapfrog Geo with implicit modelling methods. The updated geological model of the

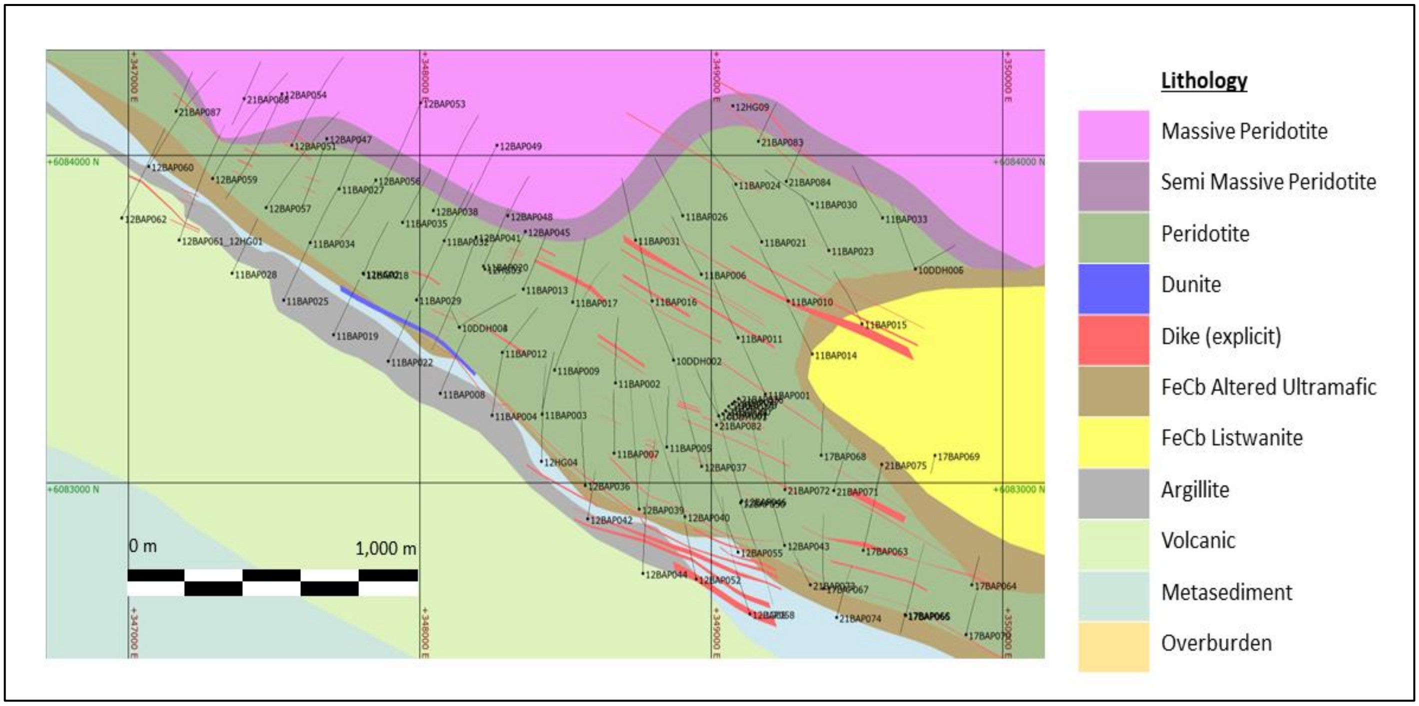
Baptiste Deposit consists of four mineralized and six unmineralized (or barren) domains (Figure 14-2). The mineralized domains include:

- Peridotite – Mineralized: main host of nickel mineralization, logged as "peridotite" (see Section 7.3.1), includes variably brecciated cataclastic and/or mylonitized peridotite.
- Dunite: grade is variable but typically lower than Peridotite – Mineralized, logged as "dunite", forms layers in Peridotite – Mineralized domain (Figure 14-2), marked by relatively fine grain size, locally brecciated.
- Peridotite – Massive: weakly mineralized, logged as "massive peridotite" or possibly "peridotite", forms transition from Peridotite – Mineralized to – Low Grade domains (Figure 14-2), more vein-controlled serpentinization as opposed to pervasive serpentinite alteration in Peridotite – Mineralized domain.
- Fe-Carbonate Altered Ultramafic: weakly mineralized, logged as "FeCb_AltUM", weak to moderate (incipient) Fe-carbonate altered peridotite, forms gradation from Peridotite – Mineralized to barren Listwanite & Fe-Cb altered domains, occurs mostly along southern margin of Baptiste Deposit and around the Listwanite ellipse bounding most of the eastern margin.

The six domains modelled as unmineralized (i.e. very low grade to barren) include:

- Peridotite – Low Grade: very low grade to barren, logged as "massive peridotite" (see Section 7.3.1), separated from Peridotite – Mineralized by Peridotite – Massive, more vein-constrained serpentinization relative to Peridotite – Mineralized domain.
- Listwanite and Fe-Carbonate Altered Ultramafic: very low grade to barren, logged as "Fe-Cb_Listwanite" and possibly the most altered examples of "FeCb_AltUM", pervasively altered peridotite consisting mostly of Fe-carbonate ± silica, separated from Peridotite – Mineralized domain by incipient Fe-Carbonate Altered Ultramafic domain.
- Altered dikes: barren, logged as "altered dikes", interpretation based on 2012 model (Ronacher et al., 2012a) extended into 2017 drilling area, sharp contacts with host rocks.
- Metavolcanic: barren, logged as volcanic, part of Sitlika succession bounding the southern margin of Baptiste Deposit, fine-grained, mafic to intermediate composition.
- Metasediments: barren, logged as argillite, part of Sitlika succession bounding the southern margin of Baptiste Deposit, fine-grained, black.
- Sitlika metasediments: barren, not intersected in any drillholes, distribution based on outcrop mapping, part of Sitlika succession, heterolithic siltstone to sandstone, variably disrupted with bedding-concordant sulphide.

Figure 14-1: Plan Map Showing Drill Locations and Lithology Model, Baptiste Deposit



Source: Next Mine, 2022.

Two key updates in the geological modelling methods were included in 2022, including:

- DTR nickel grade shells were generated to capture the zoned nature of mineralization within the peridotite lithological unit; and
- barren dikes were modeled implicitly and incorporated the dike margins as part of those intervals (as often the DTR nickel on the dike margins is low).

14.3.1.1 Grade Shells

To model the grade shells, three concentric zones were modeled, including high (>0.14% DTR nickel), medium (0.10-0.14% DTR nickel), and low grade (0.06-0.10% DTR nickel). The low-grade perimeter includes some of the FeCb altered lithologies on the margins of the mineralized peridotite. The medium grade shell is bound entirely by the low-grade shell, and nested within the medium is the high-grade shell. Each hole was interpreted and intervals within each hole were assigned the appropriate grade shell designation.

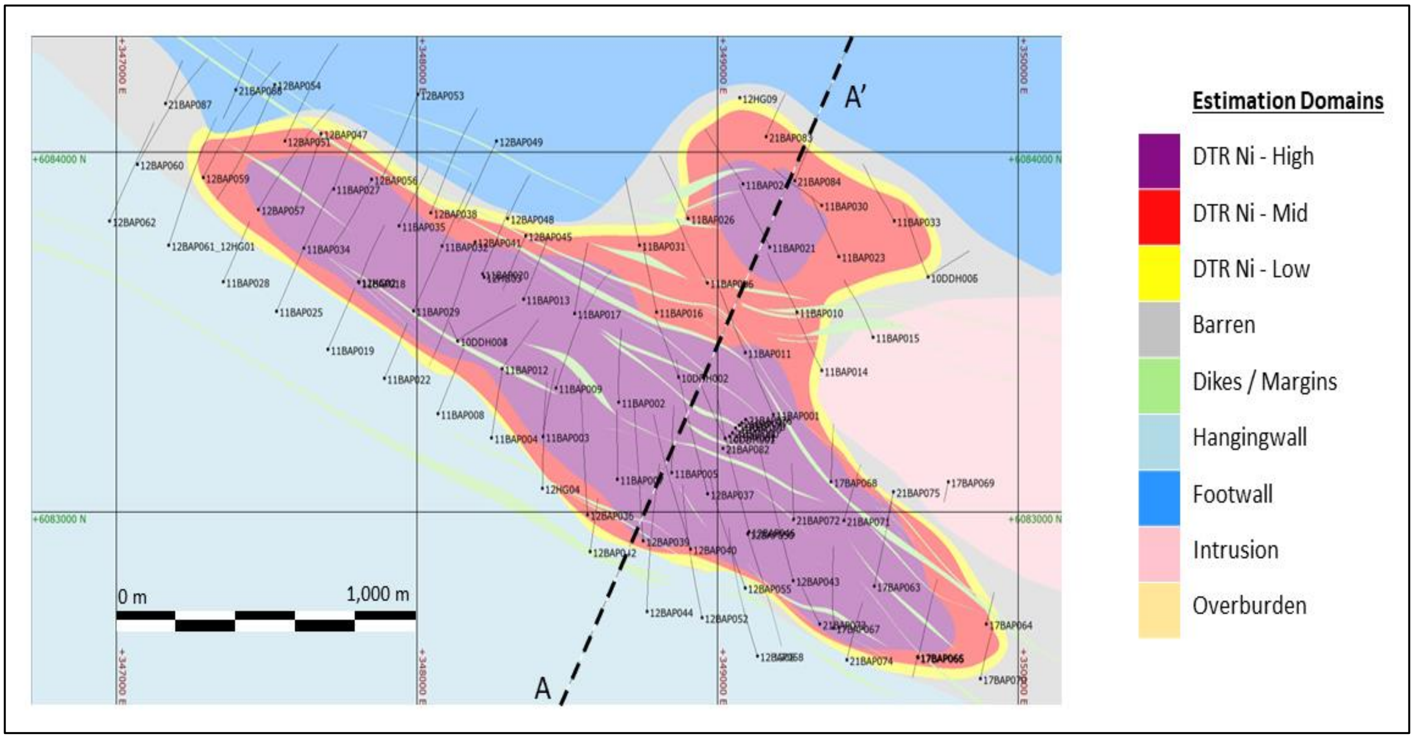
The high-grade DTR nickel domain is broken into two zones, modeled entirely within the mid-grade shell: a small zone to the north and the primary, continuous zone of high-grade mineralization to the south. The split between the two zones of high-grade mineralization is on strike and west of the 'nose' of the listwanite intrusion.

14.3.1.2 Dikes

Notably, there is an abundance of dikes following the same lineation as the split between high grade zones. Dikes were modeled as two 'swarms' that merge at the western nose of the listwanite intrusion. Two swarms associated with the North and South high-grade zones have 12 and 23 dikes modeled, respectively. Dike modelling took advantage of available dike orientations, which were taken from directional drilling data.

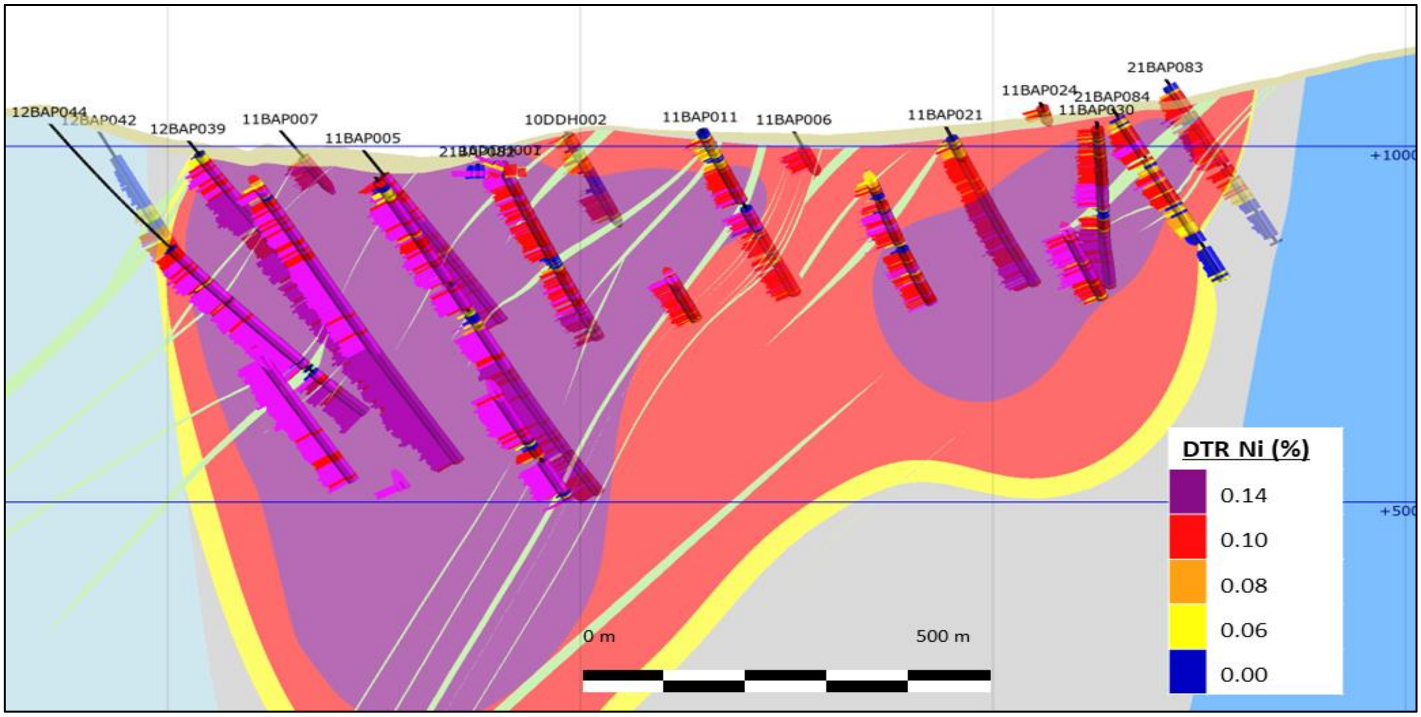
A plan view of the modeled grade shells and implicitly modeled dikes at 800 m elevation are presented in Figure 14-2, with a cross section at A-A' presented in Figure 14-3.

Figure 14-2: Plan View of Modelled Grade Shells and Implicitly Modelled Dikes at 800 m Elevation



Source: Next Mine, 2022.

Figure 14-3: Cross Section A-A' of Modelled Grade Shells and Implicitly Modelled Dikes



Source: Next Mine, 2022.

The tabulated drilling data is presented in Table 14-2 with respect to how the data contributed to the modelling of the grade shell domains, dikes, and overburden. Of the four mineralized domains (High, Mid, Low and Dikes), the high-grade domain has the largest representation of data. To read the table, for example, the high-grade domain occupies 15,514 m of drilling when the volume of that domain is evaluated against the drillholes. Of that 15,514 m, there are 15,413 m picked as high-grade in the drillhole data, 48 m as mid-grade and 53 m as dikes. Therefore, when estimating the grades for this domain in a block model, approximately 99.35% of the grades will correspond to the correct, high-grade domain, 0.3% will be mid-grade assays and 0.35% of the grades will be from intervals logged as dikes.

Table 14-2: Modelled Domains Drill Length vs Drillhole Intervals

Domain	Volume (Mm^3)	Model Length (m)	3_High (m)	2_Mid (m)	1_Low (m)	4_dikes (m)	OB (m)	Outside (m)
3_High	643	15,514	**15,413**	48	0	56	0	0
2_Mid	472	7,001	48	**6,853**	81	43	0	2
1_Low	161	1,808	0	89	**1,713**	248	0	1,585
4_dikes	168	2,729	53	10	12	**2,350**	0	54
OB	115	1,543	0	0	0	0	**1,543**	129
Outside*	8,431	5,101	0	0	2	33	0	**3,331**
Total	**9,990**	**33,695**	**15,514**	**7,001**	**1,808**	**2,729**	**1,543**	**5,101**

* All other domains that are not mineralized.

14.4 Exploratory Data Analysis

Length weighted assays within the estimated domains are presented in Table 14-3. Note that the assay lengths will not match the drillhole interval lengths as shown in Table 14-2. The discrepancy between the drilling intervals versus length of assays is significant in the Dikes domain when comparing 2,350 m of 'dike' interval vs 1,970 m of DTR Ni % assays in the database. Unassayed intervals are the primary cause of this discrepancy. For instance, where no assay value was available, intervals were assigned a code of 'dike', which is a more conservative approach than to leave the interval in one of the grade shells.

Table 14-3: Length Weighted Assays in Estimated Domains

Name	Domain	Count	Length	Mean	Min	Lower quartile	Median	Upper quartile	Max	Std. Dev.	Coeff. Var.	Variance
DTR Ni %	High	5,310	15,413	0.144	0.009	0.130	0.145	0.159	0.310	0.023	0.159	0.001
	Mid	2,623	6,945	0.109	0.003	0.096	0.109	0.123	0.245	0.022	0.204	0.000
	Low	616	1,744	0.062	0.000	0.040	0.061	0.081	0.143	0.028	0.456	0.001
	Dikes	853	1,970	0.042	0.000	0.020	0.041	0.058	0.179	0.028	0.671	0.001
DTR Fe %	High	5,310	15,413	2.47	0.22	2.02	2.47	2.91	14.12	0.64	0.26	0.41
	Mid	2,623	6,945	2.18	0.16	1.53	2.14	2.78	7.07	0.82	0.38	0.68
	Low	616	1,744	2.33	0.10	1.07	2.54	3.46	5.47	1.28	0.55	1.63
	Dikes	853	1,970	1.59	0.02	0.70	1.37	2.41	6.58	1.04	0.66	1.09
DTR Co %	High	5,187	15,204	0.0038	0.0003	0.0033	0.0038	0.0043	0.0092	0.0008	0.2138	0.0000
	Mid	2,541	6,782	0.0032	0.0003	0.0026	0.0031	0.0037	0.0069	0.0008	0.2662	0.0000
	Low	603	1,722	0.0028	0.0000	0.0015	0.0028	0.0038	0.0063	0.0014	0.5135	0.0000
	Dikes	804	1,863	0.0016	0.0001	0.0007	0.0014	0.0022	0.0050	0.0011	0.6960	0.0000
Ni ICP %	High	5,346	15,467	0.210	0.000	0.200	0.210	0.220	0.300	0.025	0.118	0.001
	Mid	2,644	6,968	0.212	0.000	0.200	0.210	0.220	0.300	0.024	0.113	0.001
	Low	637	1,793	0.209	0.000	0.200	0.210	0.227	0.270	0.033	0.158	0.001
	Dikes	1,155	2,544	0.169	0.000	0.150	0.200	0.220	0.300	0.081	0.480	0.007
Interval Length	High	5,368	-	2.89	0.00	1.00	3.00	4.00	9.73	1.27	0.44	1.60
	Mid	2,678	-	2.61	0.00	1.00	3.00	4.00	8.25	1.32	0.50	1.73
	Low	644	-	2.81	0.01	1.00	3.00	4.00	5.20	1.29	0.46	1.66
	Dikes	1,220	-	2.24	0.00	1.00	2.10	3.00	28.32	1.53	0.69	2.35

Table 14-3 also describes the interval length statistics for those assays back-flagged to the dike and grade shell domains. Mean lengths for these intervals are between 2.24 and 2.89 m (for dike and high-grade domains, respectively).

Nominal sample lengths varied from 0.12 to 9.73 m for the various drill programs, with 40% of the samples exactly 4 m in length and only 1.7% of samples exceeding 4 m in length. The targeted sample length for the 2012, 2017, and 2021 drilling campaigns was 4 m, and also matches the 1 m original plus 3 m infill sampling done on the 2011 core. The previous estimate was done using 4 m composites. However, the block z-dimension of 5 m was chosen as the composite length for the updated resource.

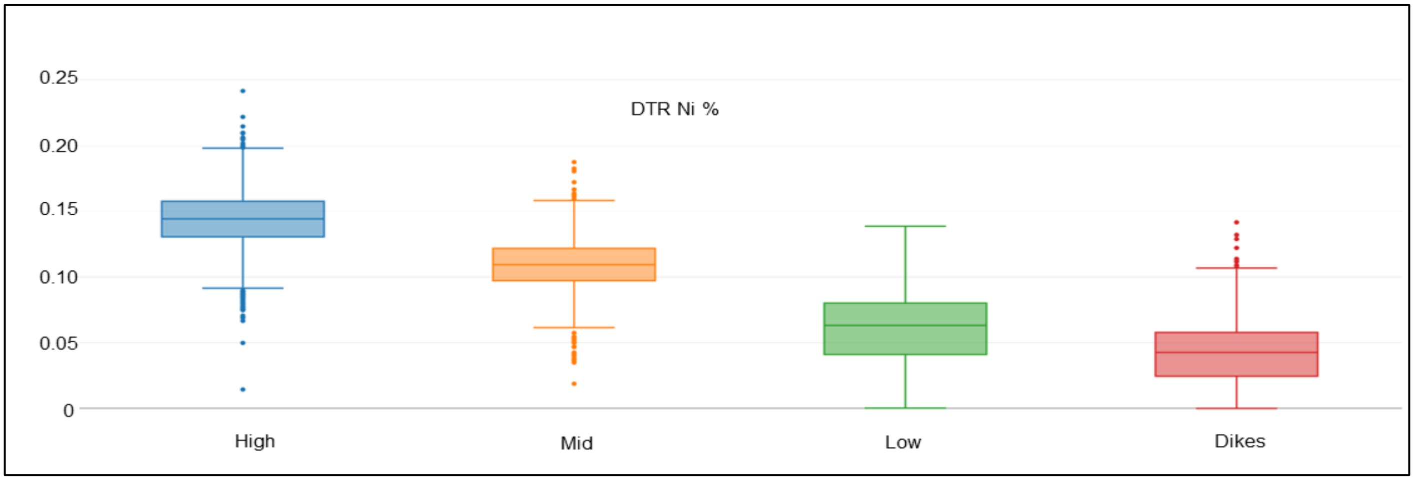
Data was therefore composited at 5 m intervals prior to statistical analysis.

Statistical analysis of the grade distribution within the mineralized domains are presented in Table 14-4, Figure 14-4, and Figure 14-5.

Table 14-4: Descriptive Statistics of 5 m Composites from Mineralized Domains

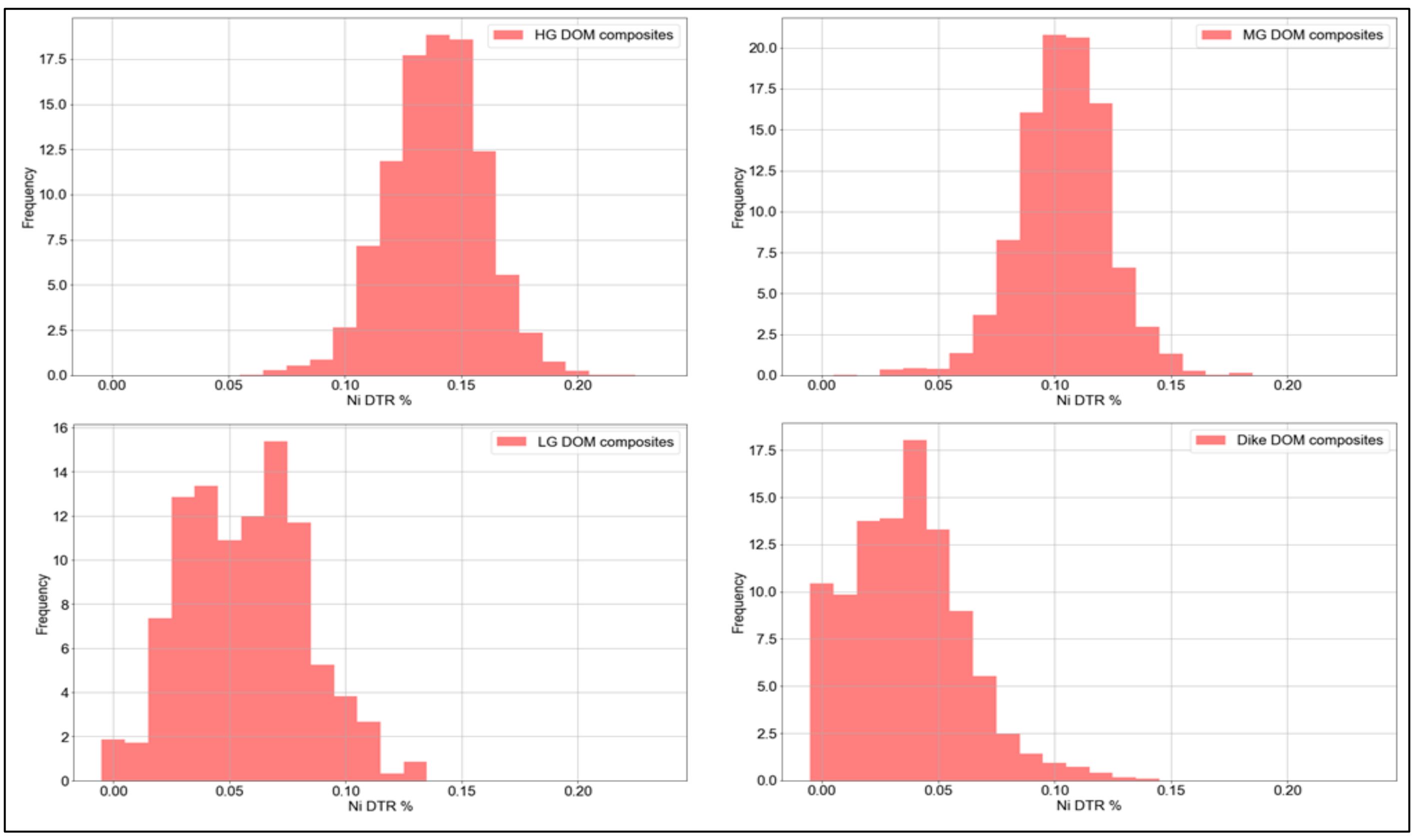
Name	Domain	Count	Length	Mean	Min	Lower quartile	Median	Upper quartile	Max	Std. Dev.	Coeff. Var.	Variance
DTR Ni %	High	3,187	15,422	0.144	0.020	0.141	0.000	0.014	0.131	0.145	0.158	0.241
	Mid	1,470	6,964	0.109	0.019	0.178	0.000	0.019	0.097	0.110	0.122	0.187
	Low	383	1,748	0.062	0.026	0.427	0.001	0.000	0.041	0.062	0.079	0.139
	Dikes	459	1,946	0.042	0.025	0.591	0.001	0.000	0.024	0.041	0.057	0.142
DTR Fe %	High	3,187	15,422	2.47	0.58	0.23	0.34	0.27	2.04	2.48	2.86	5.01
	Mid	1,470	6,964	2.18	0.77	0.35	0.60	0.53	1.56	2.18	2.74	4.85
	Low	383	1,748	2.34	1.24	0.53	1.54	0.19	1.08	2.63	3.42	4.97
	Dikes	459	1,946	1.58	0.96	0.61	0.92	0.11	0.78	1.31	2.35	4.80
DTR Co %	High	3,155	15,269	0.0038	0.0007	0.1950	0.0000	0.0003	0.0033	0.0038	0.0043	0.0072
	Mid	1,435	6,808	0.0032	0.0008	0.2392	0.0000	0.0012	0.0026	0.0031	0.0036	0.0055
	Low	379	1,731	0.0028	0.0014	0.4896	0.0000	0.0001	0.0015	0.0028	0.0039	0.0058
	Dikes	432	1,825	0.0016	0.0010	0.6467	0.0000	0.0001	0.0008	0.0014	0.0021	0.0047
Ni ICP %	High	3,196	15,467	0.210	0.020	0.093	0.000	0.010	0.199	0.210	0.221	0.290
	Mid	1,473	6,979	0.212	0.018	0.087	0.000	0.049	0.202	0.212	0.222	0.290
	Low	391	1,786	0.209	0.027	0.131	0.001	0.005	0.203	0.212	0.224	0.256
	Dikes	587	2,477	0.172	0.070	0.408	0.005	0.000	0.136	0.200	0.220	0.300
Interval Length (m)	High	3,196	-	4.84	0.70	0.15	0.49	0.02	5.00	5.00	5.00	5.00
	Mid	1,473	-	4.74	0.88	0.19	0.78	0.03	5.00	5.00	5.00	5.00
	Low	391	-	4.57	1.12	0.25	1.25	0.02	5.00	5.00	5.00	5.00
	Dikes	587	-	4.22	1.34	0.32	1.79	0.00	3.96	5.00	5.00	5.00

Figure 14-4: Box Plots Showing the % DTR Nickel Grade Distribution Within Mineralized Domains



Source: Next Mine, 2022.

Figure 14-5: Frequency Distribution of % DTR Nickel Grade Within Mineralized Domains



Source: Next Mine, 2022.

14.5 Density Assignment

A total of 978 specific gravity measurements were used to assign bulk density values to the lithologic domains. The median value of the readings for each rock type were used, except in the case of the metavolcanics where the single result of 2.23 was deemed unreliable, so 2.71 was assigned.

The median density for the four ultramafic domains ranges from 2.7 to 2.8 g/ cm^3, which is closer to the typical densities for serpentine (2.2-2.9 g/cm^3) than it is to olivine (3.2-4.5g/ cm^3). Median domain densities are therefore consistent with the pervasive serpentinization of ultramafic rocks of the Baptiste Deposit.

Table 14-5: Density Assignments for Each Lithological Domain

Lithological Domain	Previous		Current	
	Specific Gravity Measurements	Median Density (g/cm^3)	Specific Gravity Measurements	Median Density (g/cm^3)
Dunite	55	2.66	79	2.66
Peridotite – Mineralized	638	2.67	764	2.67
Peridotite – Massive	102	2.79	151	2.78
Fe-Carbonate Altered Ultramafic	57	2.68	53	2.66
Listwanite and Fe-Carbonate Altered Ultramafic	65	2.69	71	2.67
Metavolcanic	1	2.71*	1	2.71*
Argillite, Sitlika Metasediment	7	2.71	7	2.71
Altered Dikes	53	2.92	76	2.86
Total	**978**	**-**	**1,202**	**-**

* Actual measured density was 2.23 g/cm^3 but is considered unrepresentative.

14.6 Grade Capping and Outlier Restrictions

Grade distribution in the composited data was examined to determine if grade capping or special treatment of high outliers was warranted. Cumulative probability plots were examined for outlier populations and decile analyses was performed for DTR nickel within the mineralized domains. Generally, the cutting or restriction of high grades is warranted if the last decile (upper 10% of samples) contains >40% of the metal or more than 2.3 times the metal of the previous decile, or if the last centile (upper 1%) contains >10% of the metal or >1.75 times the metal of the next highest centile. For the H, M, and L grade shell domains, the last decile contained only 14% of the metal and the last centile contained 1.5% (Table 14-6); the conclusion is no capping or restriction of composites is warranted.

Table 14-6: Decile Analysis (0-100) for Mineralized Domains

	Decile	Count	DTR Ni Grade			Contained	Contained Percentage
			Average (%)	Min (%)	Max (%)		
DTR Ni	0-10	504	0.061	0.000	0.086	1.40	4.5%
	10-20	504	0.096	0.086	0.104	2.26	7.3%
	20-30	504	0.109	0.104	0.115	2.62	8.5%
	30-40	504	0.119	0.115	0.123	2.86	9.2%
	40-50	505	0.127	0.123	0.131	3.14	10.2%
	50-60	503	0.135	0.131	0.139	3.27	10.6%
	60-70	504	0.142	0.139	0.146	3.46	11.2%
	70-80	504	0.150	0.146	0.154	3.69	11.9%
	80-90	504	0.159	0.154	0.164	3.95	12.8%
	90-100	504	0.174	0.164	0.241	4.29	13.9%
	Total	**5,040**	**0.127**	**0.000**	**0.241**	**30.92**	**-**

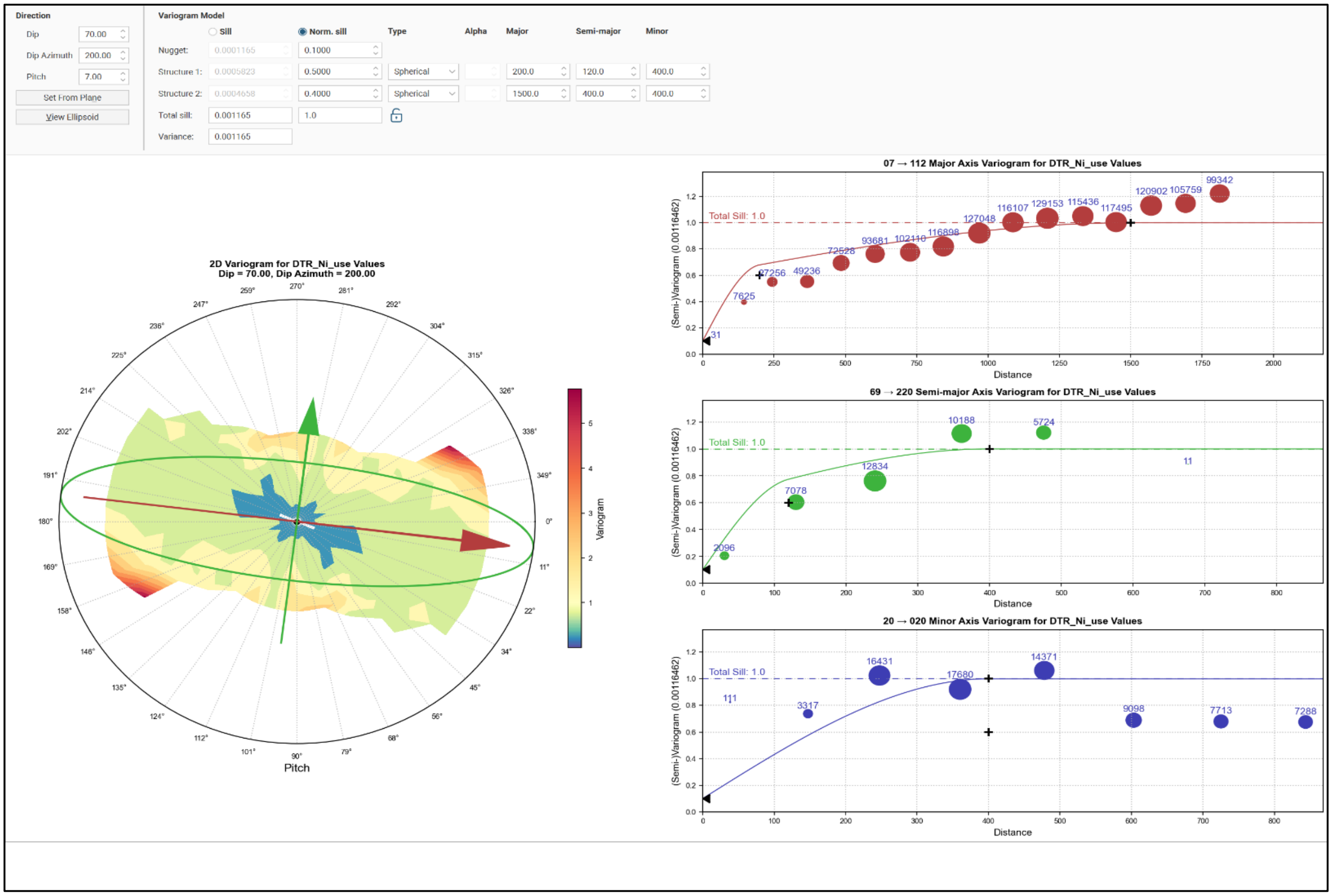
Table 14-7: Centile Analysis (90-100) for Mineralized Domains

	Centile	Count	DTR Ni Grade			Contained	Contained Percentage
			Avg (%)	Min (%)	Max (%)		
DTR Ni	90-91	51	0.165	0.164	0.165	0.40	1.3%
	91-92	50	0.166	0.166	0.167	0.41	1.3%
	92-93	51	0.168	0.167	0.168	0.42	1.4%
	93-94	50	0.169	0.168	0.170	0.41	1.3%
	94-95	50	0.170	0.170	0.172	0.42	1.4%
	95-96	51	0.173	0.172	0.174	0.43	1.4%
	96-97	50	0.175	0.174	0.177	0.43	1.4%
	97-98	51	0.179	0.177	0.181	0.45	1.4%
	98-99	50	0.184	0.181	0.186	0.44	1.4%
	99-100	50	0.197	0.187	0.241	0.47	1.5%
	Total	**504**	**0.174**	**0.164**	**0.241**	**4.29**	**-**

14.7 Variography

Kriging parameters, search parameters, and anisotropy were determined with semi-variograms for percent DTR nickel using composites falling within the mineralized domains. Models showed a maximum range of 1,500 m (Figure 14-6). The resulting search ellipsoid has a major axis trending at an azimuth of 110° with a near-vertical semi-major axis.

Figure 14-6: Variogram Models for Different Plunges and Azimuths of Search Windows



Source: Next Mine, 2022.

14.8 Estimation/Interpolation Methods

A block model was created in Leapfrog Edge software (version 2021.2.4) using a block size with dimensions of 10 m x 10 m x 5 m. Model extents are presented in Table 14-8.

Table 14-8: Block Model Extents

	Easting (NAD83, Zone 10)	Northing (NAD83, Zone 10)	Elevation (masl)
Minimum	346,170	6,083,495	200
Block size (m)	10	10	5
Blocks (quantity)	422	250	288
Sub-blocks (quantity)	4	4	2
Azimuth	27.5 degrees (rotate clockwise around the Z axis when looking down)		
Dip	0 degrees (then rotate around the X' axis down from the horizontal plane)		
Pitch	0 degrees (then rotate clockwise around the Z'' axis when looking down)		

Parent blocks were sub-blocked to achieve precise volumes for domains, overburden contacts, and the topography contact. Parent blocks were sub-blocked to a minimum of 2.5 m in all dimensions. The geological model (GM) volumes of estimation domains are represented by block volumes with a variance of less than 0.01% for all estimated domains, as presented in Table 14-9.

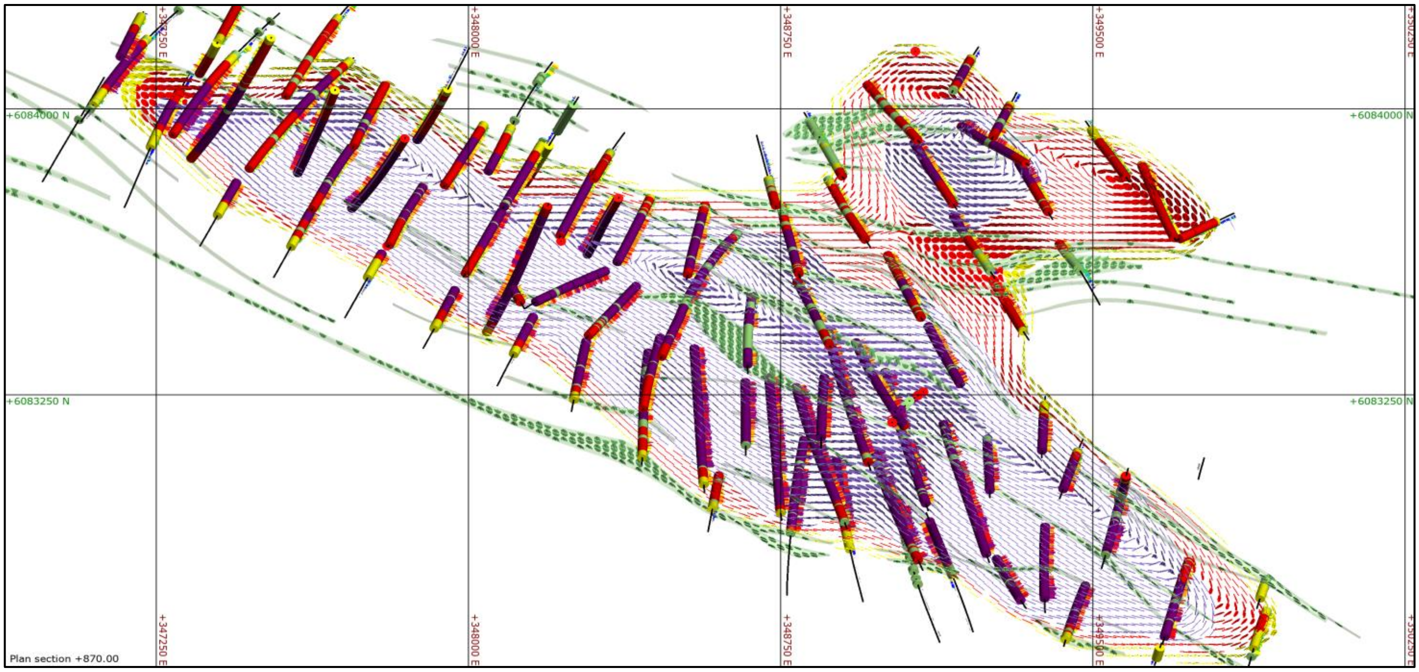
Table 14-9: Domain Wireframe Volume vs Block Model Volume

Domain	GM Volume (million m³)	Block Volume (million m³)	Variance (%) (block – GM)
dikes	66.59	66.60	0.02
grade shell High	488.81	488.86	0.01
grade shell Mid	250.64	250.86	0.09
grade shell Low	45.17	45.20	0.07
Total	**851.21**	**851.53**	**0.04**

14.8.1 Estimation Methodology

DTR nickel grades within the corresponding domains were estimated in two passes using ordinary kriging (OK) for grade shells and inverse distance squared (ID2) for the dikes. A single pass nearest-neighbour (NN) estimate was used to decluster composites for use in model validation swath plots. The anisotropy conforms to the search ellipsoids derived from the variogram models. Locally varying anisotropy was used for all grade shell and dike domains. The dikes utilized the central trend line derived from the Leapfrog 3D vein models. The grade shells employed concentric patterns that honoured the shape of their 'nested' contacts and that samples should be paired as such for estimation. The high-grade shell also used the major axis of the variogram model. The variable orientation azimuth and dip ellipsoids are presented in Figure 14-7.

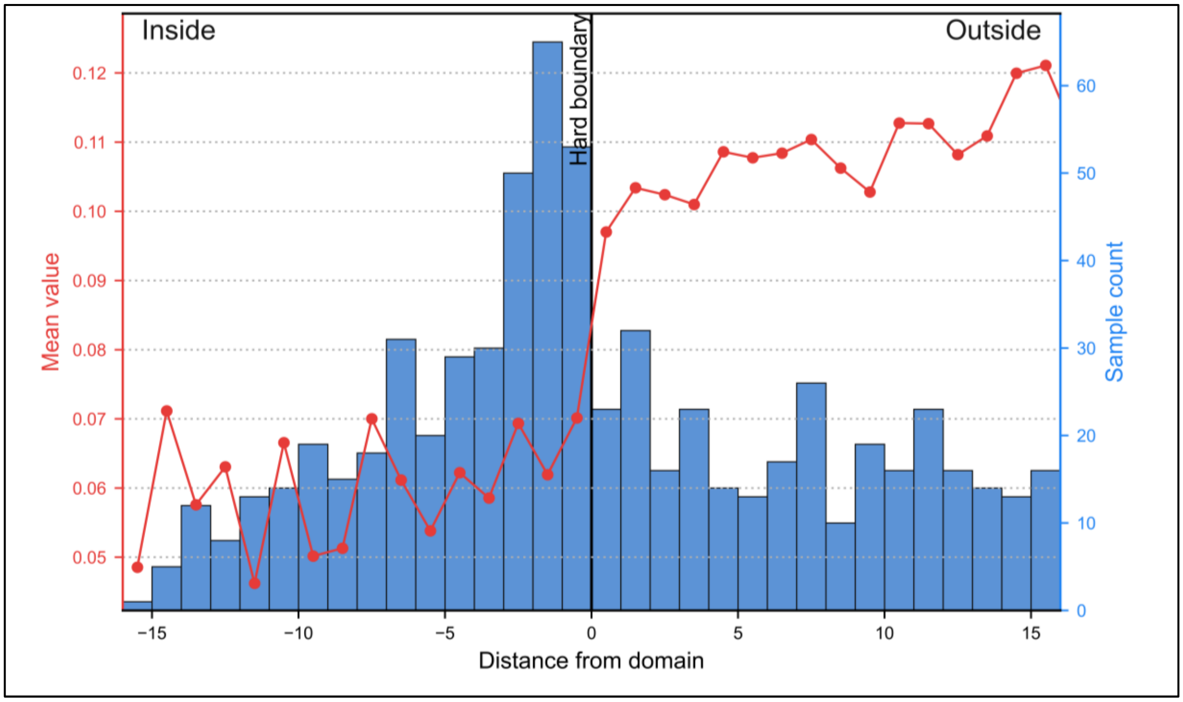
Figure 14-7: Variogram Models for Different Plunges and Azimuths of Search Windows



Source: Next Mine, 2022.

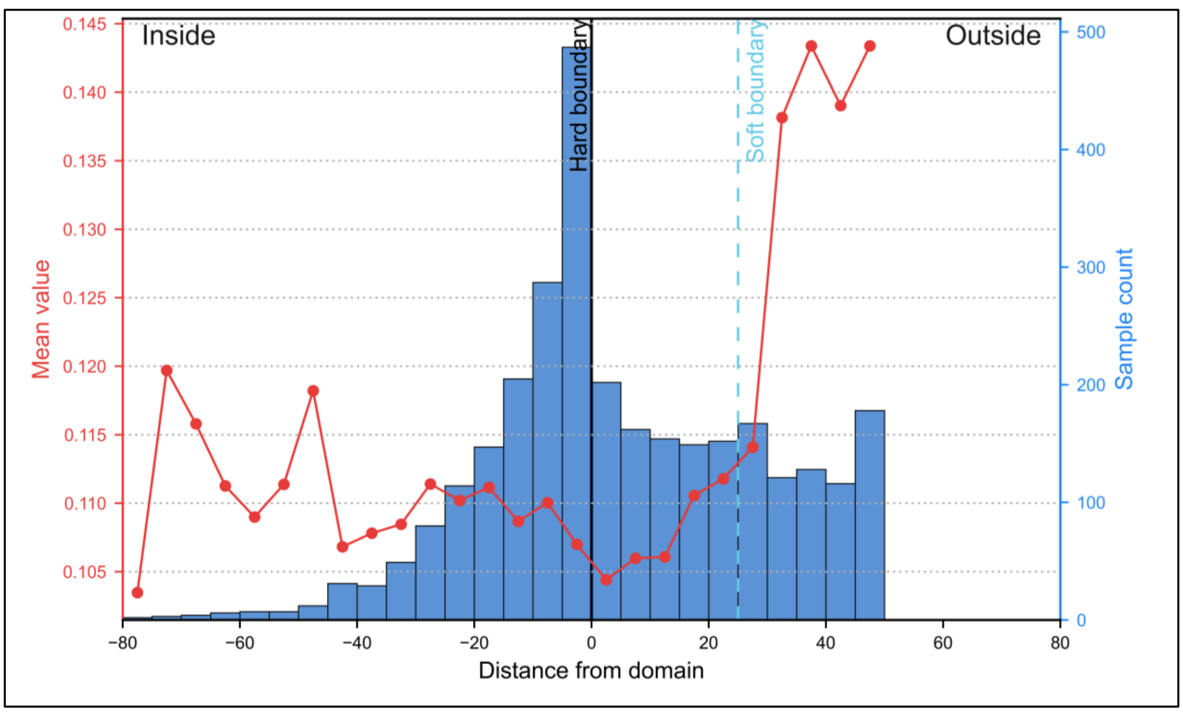
Semi-soft boundaries (± 25 m) were used between the High and Mid sub-domains as these contacts are gradational. A hard boundary was used at the Low sub-domain because that domain is largely within a separate lithological unit. A hard boundary was also used between the peridotite grade shells and the dike domains.

Figure 14-8: % DTR Nickel Contact Plot of Low-Grade Domain



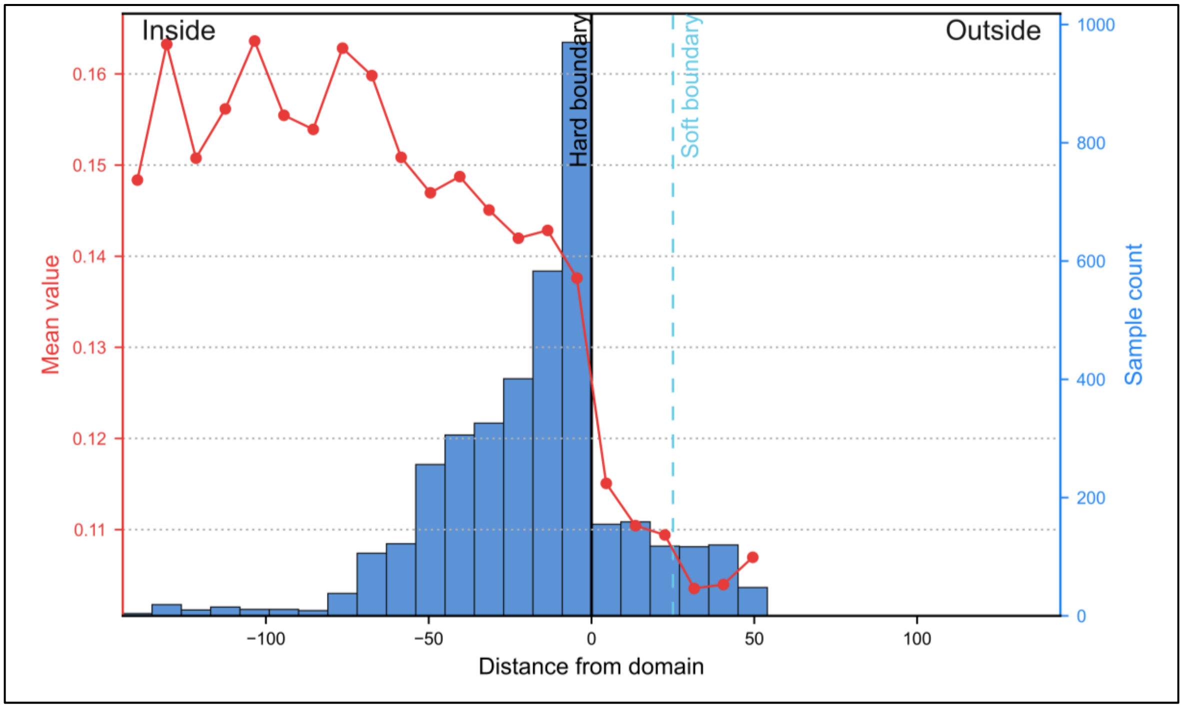
Source: Next Mine, 2022.

Figure 14-9: % DTR Nickel Contact Plot of Mid-Grade Domain



Source: Next Mine, 2022.

Figure 14-10: % DTR Nickel Contact Plot of High-Grade Domain



Source: Next Mine, 2022.

Mineralized domains of the Baptiste Deposit are cut by 33 steeply dipping, non-mineralized dikes, comprising approximately 12% (formerly was 3% in 2020 Resource Estimate) of the rock mass in the classified resource blocks. These dikes are of varying thickness and can be identified as rock units that could be selectively mined as waste.

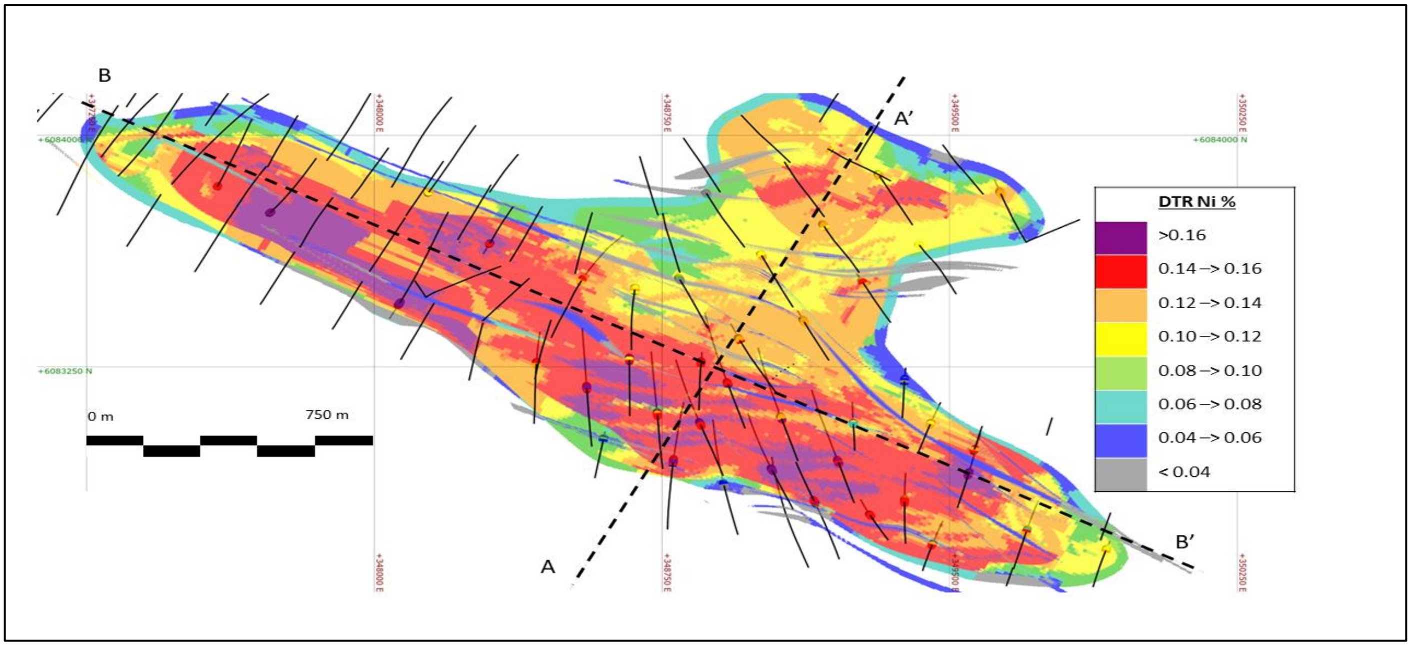
Search parameters as well as all estimation parameters are presented in Table 14-10.

Table 14-10: Grade Model Search Parameters

Variable	Domain	Value Filter	Boundary, Range (m)	Estimation Method	Variable Orientation	Pass	Search Max Dir (m)	Search Inter Dir (m)	Search Min Dir (m)	Min No. Samples	Max No. Samples	Max Samples per Hole
DTR Ni %	Low	M, L	Hard	OK	1_L_VO	i	200	120	120	3	8	2
						ii	999	750	500	1	7	
	Mid	H, M, L	SS, 25	OK	2_M_VO	i	200	120	120	3	8	2
						ii	999	750	500	1	7	
	High	H, M	Hard	OK	3_H_VO	i	200	120	120	3	8	2
						ii	999	750	500	1	7	
	Dikes	Dikes	Hard	ID	Dike_VO	i	607	364	121	12	36	8
Ni ICP %	Low	M, L	SS, 40	OK	1_L_VO	ii	607	364	121	1	36	
						i	200	120	120	3	8	2
	Mid	H, M, L	SS, 50	OK	2_M_VO	ii	999	750	500	1	7	
						i	200	120	120	3	8	2
	High	Dikes	Hard	OK	3_H_VO	ii	999	750	500	1	7	
						i	200	120	120	3	8	2
	Dikes	H, M, L	Hard	ID	Dike_VO	ii	999	750	500	1	7	
						i	607	364	121	12	36	8
DTR Co %	All	Dikes	Hard	OK	L_VO	ii	607	364	121	1	36	
						i	200	120	120	3	8	2
	Dikes	H, M, L	Hard	ID	Dike_VO	ii	999	750	500	1	7	
						i	607	364	121	12	36	8
DTR Fe %	All	Dikes	Hard	OK	L_VO	ii	607	364	121	1	36	
						i	200	120	120	3	8	2
						ii	999	750	500	1	7	
	Dikes	Dikes	Hard	ID	Dike_VO	i	607	364	121	12	36	8
						ii	**607**	**364**	**121**	**1**	**36**	

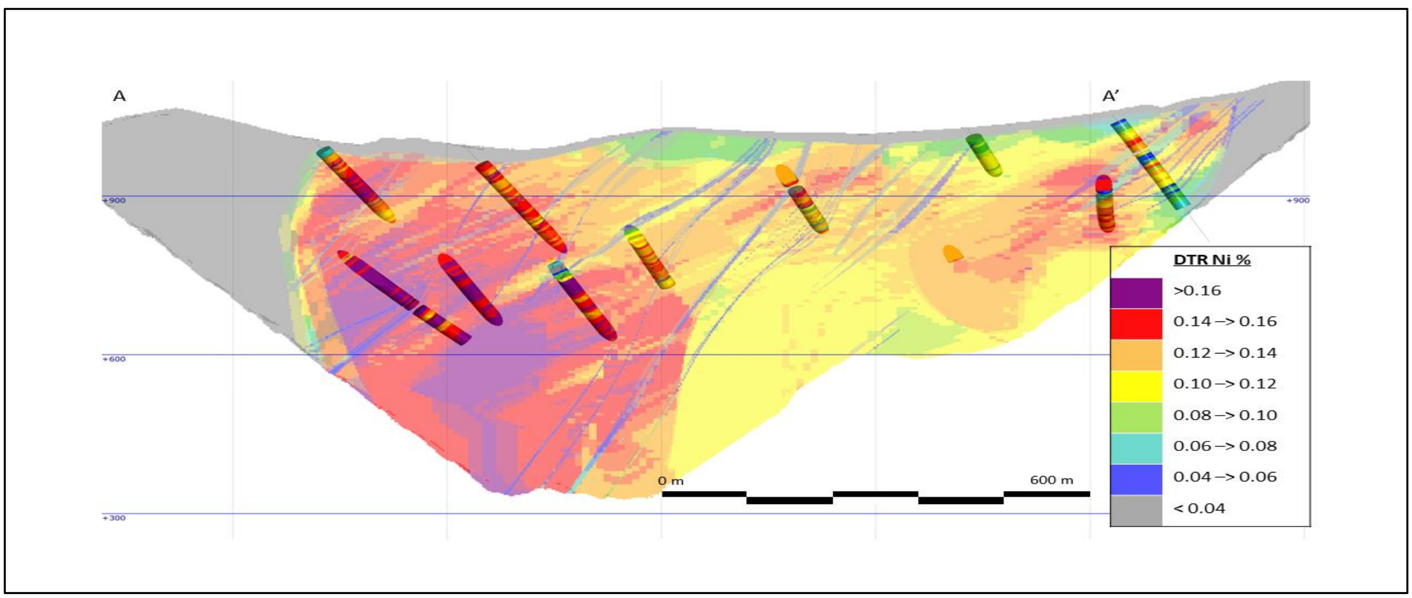
Block model grade distribution is presented in Figure 14-12 to Figure 14-14.

Figure 14-11: % DTR Nickel Block Grade Distribution in Plan View at 800 m



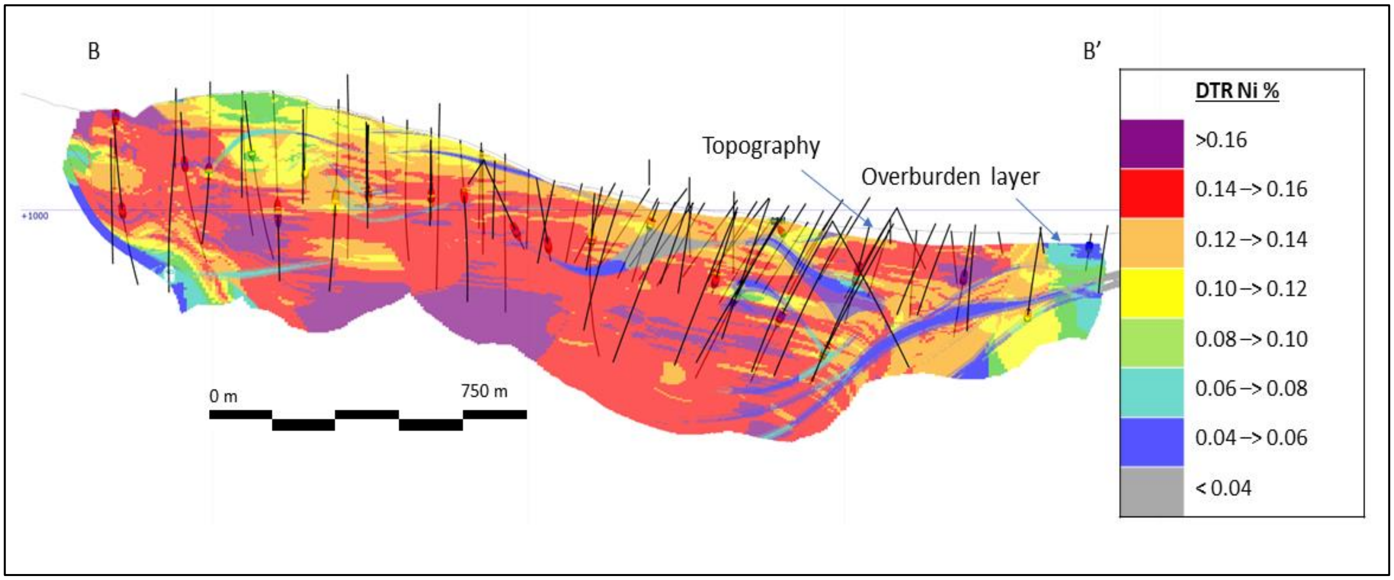
Source: Next Mine, 2022.

Figure 14-12: Block Grade Distribution on Cross Section A – A' from Figure 14-11



Source: Next Mine, 2022.

Figure 14-13: Block Grade Distribution on Longitudinal Section B – B' from Figure 14-11



Source: Next Mine, 2022.

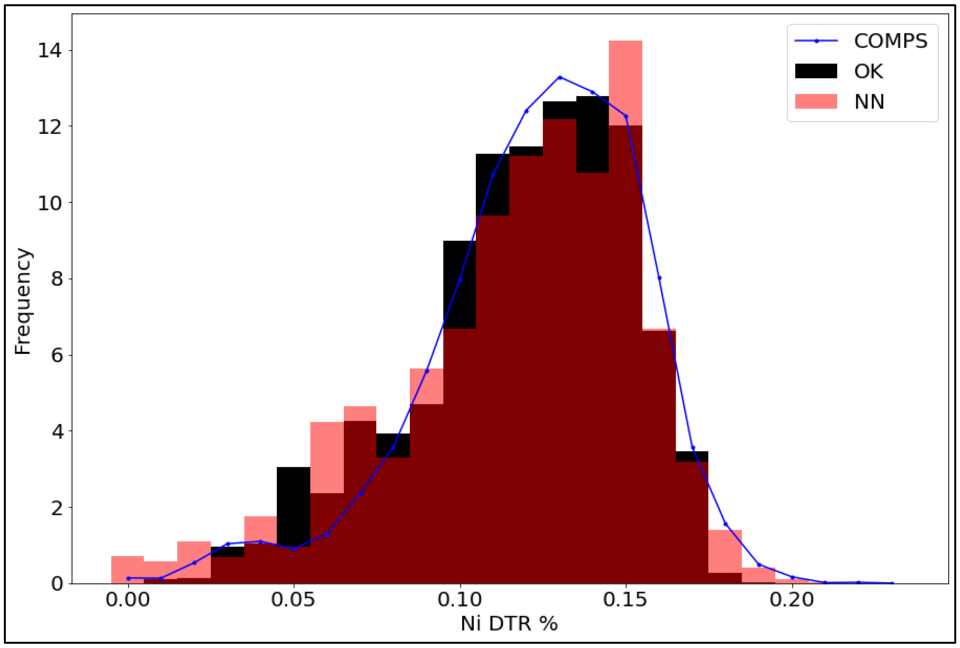
14.9 Block Model Validation

Block model validation included visual inspection, global bias check (Figure 14-14), and a check for local bias. Each of these is summarized below.

Visual inspection comprised a visual comparison of blocks and composite grades in plan and section views. The estimated block grades showed reasonable correlation with adjacent composite grades.

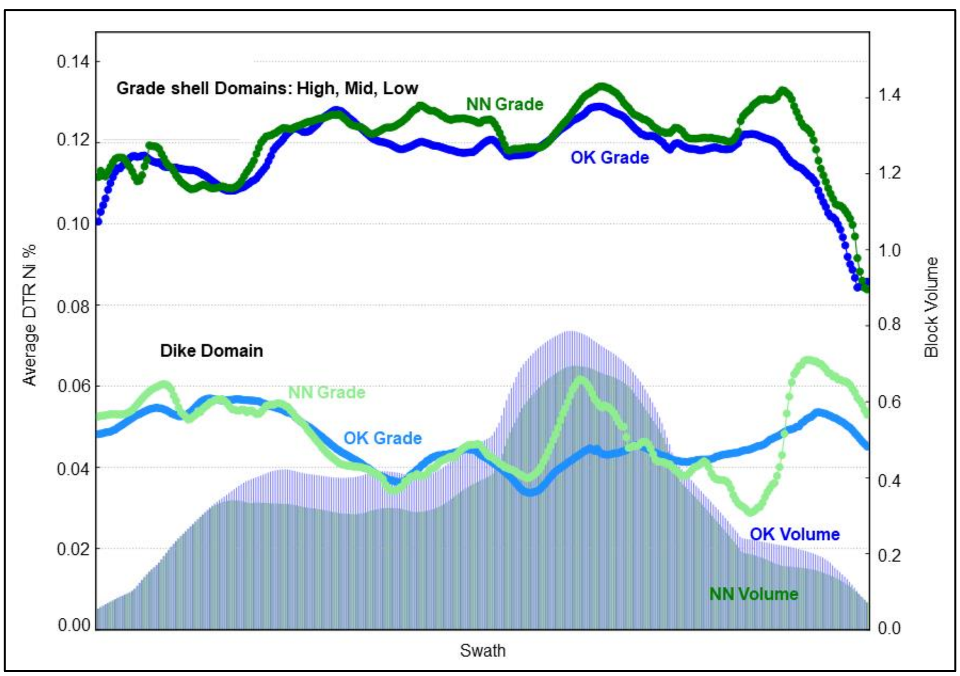
The local bias check was done with swath plots that were generated to compare OK, ID2 and nearest-neighbour estimates on panels through the Deposit. Results show a reasonable comparison between the methods, as indicated by the bar charts presented in Figure 14-15 to Figure 14-17, particularly in the main portions of the deposit.

Figure 14-14: Block Grade Distribution for the Baptiste Deposit Sub-Domains High, Mid, Low



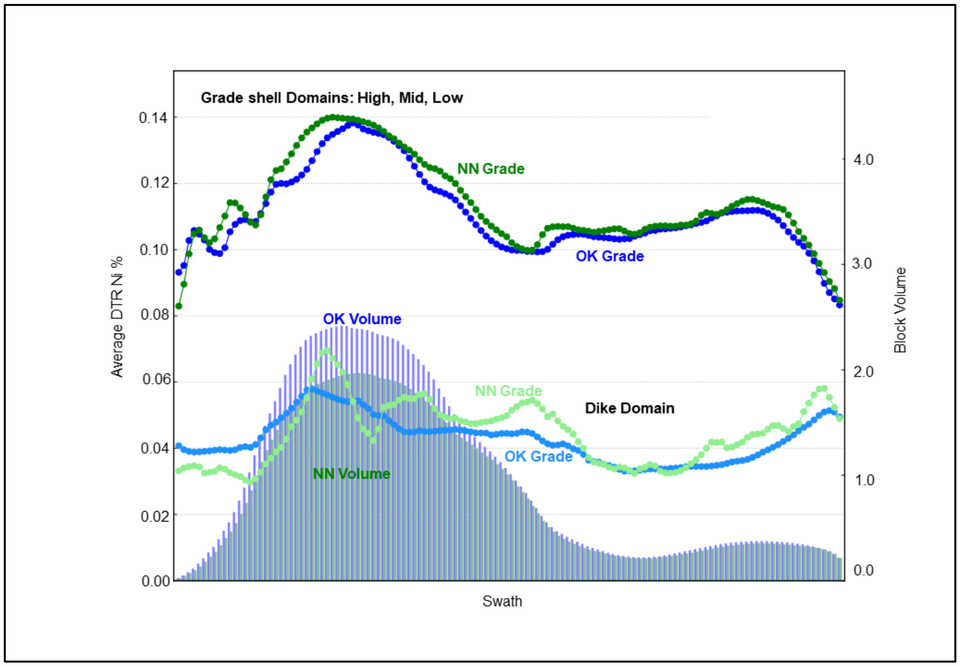
Source: Next Mine, 2022.

Figure 14-15: NNE – SSW Trending Swath Plot (X-Direction)



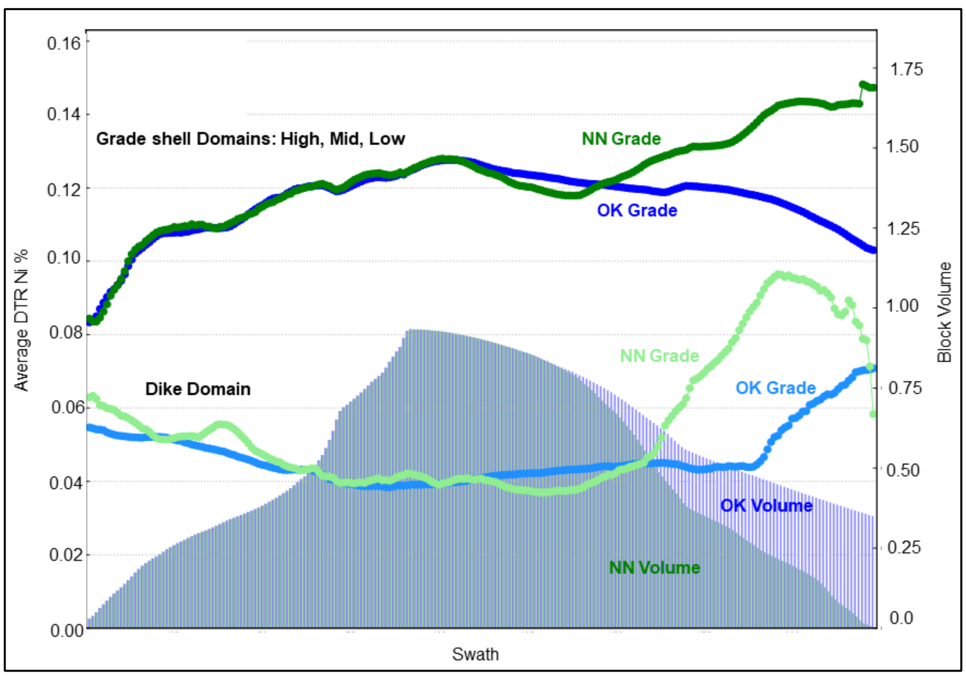
Source: Next Mine, 2022.

Figure 14-16: ESE – WNW Oriented Swath Plot (Y-Direction)



Source: Next Mine, 2022.

Figure 14-17: Elevation Oriented Swath Plot (Z-Direction)



Source: Next Mine, 2022.

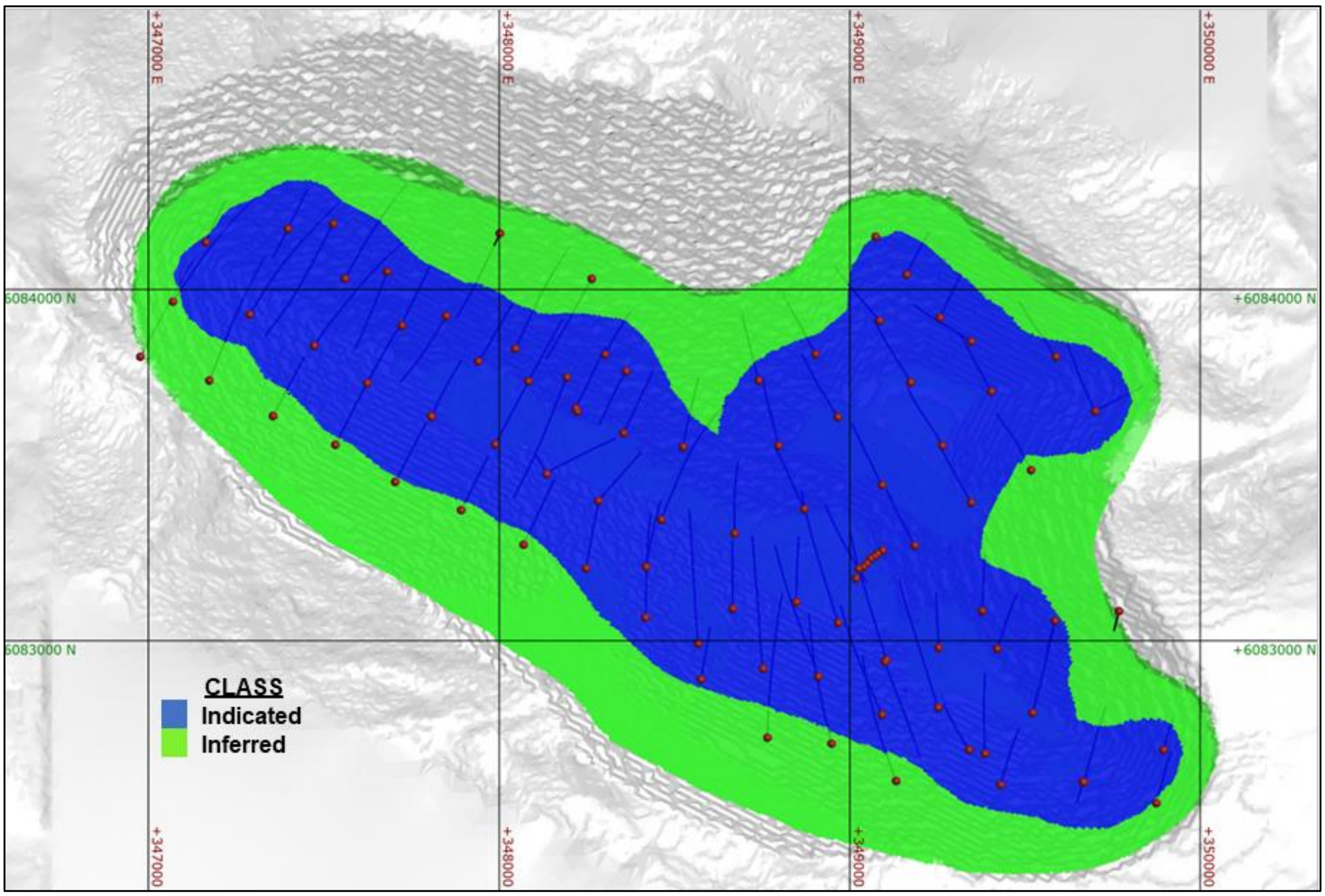
14.10 Classification of Mineral Resources

Resource classifications used in this study conform to the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014). To be classified as an indicated mineral resource a block had to meet the following conditions:

- estimated using composites from three separate holes and restricted to one of the four mineralized sub-domains (Grade shell: High, Mid, Low and or in the Dike domain); and
- within a 200 m drill spacing.

Blocks not classified as indicated mineral resources were assigned to the inferred mineral resource category if they fell within a 300 m drill spacing and used a minimum of two holes (Figure 14-18).

Figure 14-18: Plan View of Resource Classification with 2022 Resource Shell



Source: Next Mine, 2022.

14.11 Reasonable Prospects for Eventual Economic Extraction

Mineral resources were constrained by an optimized pit shell based on an exchange rate of C$1 = US$0.77, a nickel price of US$8.50/lb, and 96% payability. Total operating costs, including mining, processing, and general and administrative, plus minimum profit was assumed at US$9.37/t milled. A mining recovery of 95% and process recovery of 85% were used in the optimization. The pit slopes considered were 44˚ on the south wall and 42˚ on the north wall.

A base case cut-off grade of 0.06% DTR Ni represents an in-situ metal value of approximately US$9/t, which is believed to provide a reasonable margin over operating and sustaining costs for open pit mining and processing.

14.12 Mineral Resource Statement

Table 14-11 provides the summary of the 2022 Baptiste Deposit pit-constrained Mineral Resource Estimate.

Table 14-11: 2022 Baptiste Deposit Pit-Constrained Mineral Resource Estimate*

Category	Tonnes (Mt)	DTR Ni content		Total Ni content		DTR Fe content		DTR Co content	
		(% Ni)	(kt Ni)	(% Ni)	(kt Ni)	(% Fe)	(kt Fe)	(% Co)	(kt Co)
Indicated	1,815	0.129	2,435	0.211	3,828	2.40	43,535	0.0035	64.4
Inferred	339	0.131	2,436	0.212	720	2.55	8,634	0.0037	12.5

Notes:
1. Mineral Resource Estimate prepared by Richard Flynn, P. Geo., of Next Mine Consulting using ordinary kriging within grade shell domains and inverse distance squared in dike domains; effective date of November 14, 2022.
2. Resources are reported using the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines.
3. Davis Tube magnetically-recovered (DTR) nickel is the nickel content recovered by magnetic separation using a Davis Tube, followed by Fusion XRF to determine the nickel content of the magnetic fraction; in effect a mini-scale metallurgical test. The Davis Tube method is the global, industry standard metallurgical testing apparatus for recovery of magnetic minerals.
4. Indicated mineral resources are drilled on approximately 200 x 200 metre drill spacing and confined to mineralized lithologic domains. Inferred mineral resources are drilled on approximately 300 x 300 metre drill spacing.
5. A cut-off grade of 0.06% DTR Ni was applied.
6. An optimized pit shell was generated using the following assumptions: US$8.50/lb nickel Price; pit slopes between 42-44º; nickel payability of 96%, mining recovery of 97% DTR Ni, process recovery of 85% DTR Ni, exchange rate of US$1.00=C $0.77; and total operating cost and minimum profit of US$9.37/t milled.
7. Totals may not sum due to rounding.
8. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

The effective date of the 2020 mineral resource estimate is November 14, 2022. See Table 14-11 for additional notes concerning preparation of the mineral resource estimate.

It is reasonably expected that most of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. Table 14-12 presents the Mineral Resource Estimate for the Baptiste Deposit at a range of cut-off grades with the Base Case, an applied cut-off grade of 0.06% DTR Nickel noted in bold face.

Table 14-12: Indicated and Inferred Resources for the Baptiste Deposit

INDICATED			
% DTR Ni Cut-off	**Tonnes (Mt)**	**DTR Ni (%)**	**Contained Ni (kt)**
0.04	1,850	0.128	2,364
0.06	**1,815**	**0.129**	**2,346**
0.08	1,730	0.132	2,285
INFERRED			
% DTR Ni Cut-off	**Tonnes (Mt)**	**DTR Ni (%)**	**Contained Ni (kt)**
0.04	351	0.128	449
0.06	**339**	**0.131**	**443**
0.08	319	0.134	428
0.1	288	0.139	400

Notes:

1. Mineral Resource Estimate prepared by Richard Flynn, P.Geo of Next Mine Consulting using ordinary kriging within grade shell domains and inverse distance squared in dike domains; effective date of November 14, 2022.
2. Resources are reported using the 2014 CIM Definition Standards and were estimated in accordance with the CIM 2019 Best Practices Guidelines.
3. Davis Tube magnetically-recovered (DTR) nickel is the nickel content recovered by magnetic separation using a Davis Tube, followed by Fusion XRF to determine the nickel content of the magnetic fraction; in effect a mini-scale metallurgical test. The Davis Tube method is the global, industry standard metallurgical testing apparatus for recovery of magnetic minerals.
4. Indicated mineral resources are drilled on approximately 200 x 200 m drill spacing and confined to mineralized lithologic domains. Inferred mineral resources are drilled on approximately 300 x 300 m drill spacing.
5. A cut-off grade of 0.06% DTR Ni was applied.
6. An optimized pit shell was generated using the following assumptions: US\$8.50/lb nickel Price; pit slopes between 42-44^{o}; nickel payability of 96%, mining recovery of 97% DTR Ni, process recovery of 85% DTR Ni, exchange rate of US\$1.00=C \$0.77; and total operating cost and minimum profit of US\$9.37/t milled.
7. Totals may not sum due to rounding.
8. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

14.13 Comparison with the Previous Mineral Resource Estimate

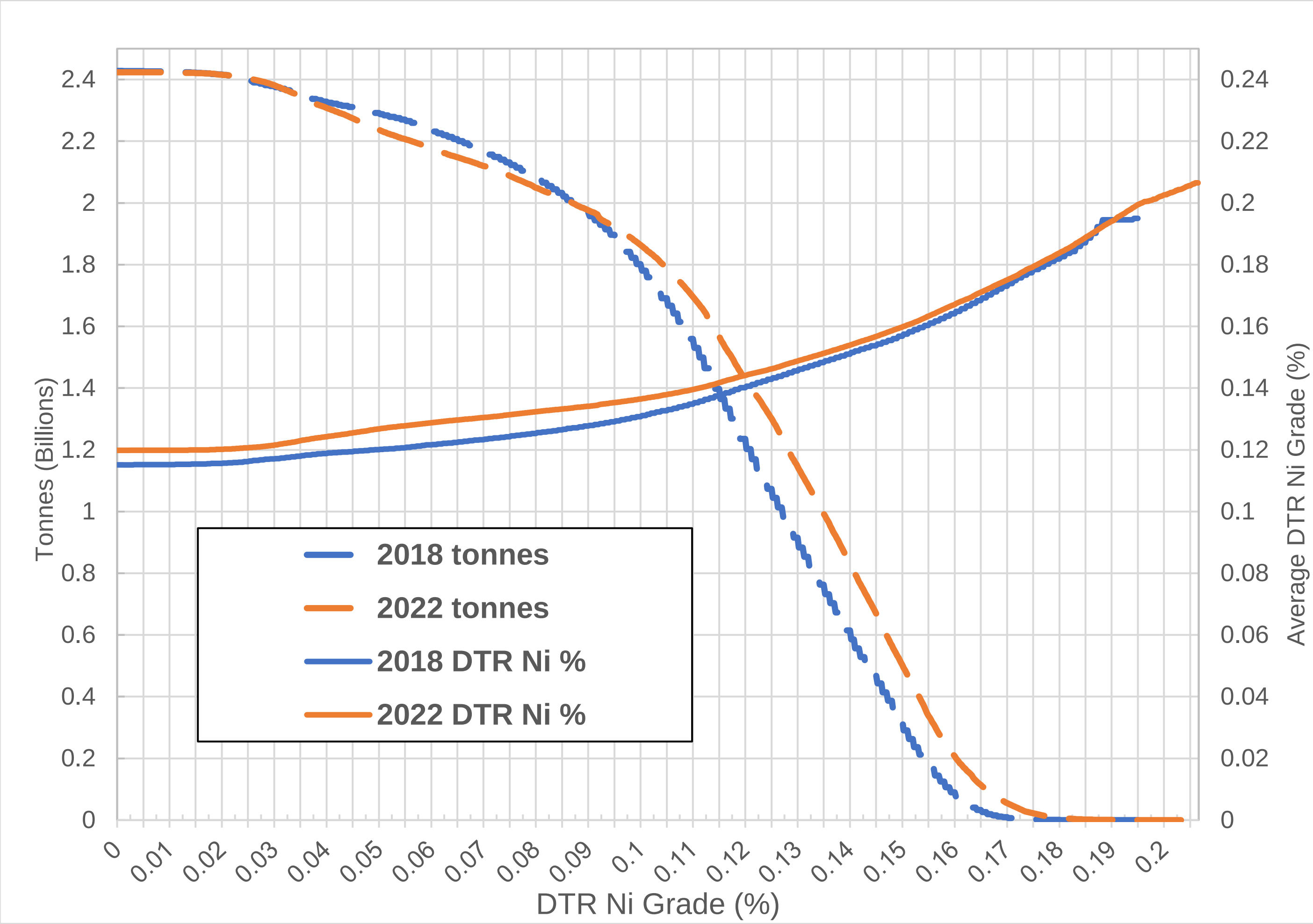
Table 14-13 presents a comparison between the 2018 and 2022 Baptiste Deposit pit-constrained Mineral Resource Estimate.

Table 14-13: Comparison of 2018 vs 2022 Indicated and Inferred Resources

Category	2018			2022			DELTA		
	Tonnes (Mt)	DTR Ni content		Tonnes (Mt)	DTR Ni content		Tonnes (Mt)	DTR Ni content	
		(% Ni)	(kt Ni)		(% Ni)	(kt Ni)		(% Ni)	(kt Ni)
Indicated	1,843	0.123	2,271	1,815	0.129	2,346	-2%	5%	3%
Inferred	391	0.115	448	339	0.131	443	-13%	14%	-1%

Figure 14-19 presents the Grade – Tonnage curve comparison between the 2018 and 2022 Baptiste Deposit resource estimates.

Figure 14-19: Grade – Tonnage Curves* Comparing 2018 and 2022 Resource Estimates



Note: Blocks were restricted to Indicated and Inferred blocks within each year's respective resource pit shell which are optimized for the block models.
Source: Next Mine, 2022.

When comparing the estimates from 2018 and 2022, there are a few key differences. For both Indicated and Inferred resources, the increased grade and reduced tonnage are an outcome from the efforts made to improve geological modelling and interpretation at Baptiste. Primary improvements are outlined earlier in Section 14, but are primarily due to the implicit modelling of the dikes and the addition of DTR nickel grade shells to sub-domain the mineralized peridotite lithological unit.

Mean DTR Ni grades were compared within mineralized domains in the data prior to 2021 with the data gathered in the 2021 bore hole assays. There is no change; both data sets share a mean DTR Ni of 0.110 %. Therefore, the improved grade in the 2022 resource estimate is attributable to the advanced geological interpretation of the mineralized peridotite.

Of note between the 2018 and 2022 resource estimates is the DTR Ni grade of Inferred resources. The 2022 grade is much higher; higher than the grade of Indicated resources. This discrepancy is uncommon in resource estimates where typically, the grade of Inferred resources is less than Indicated resources. However, as seen in the z–direction swath plots, the grade of DTR Ni increases sharply at depth, and at depth the blocks are categorized as Inferred.

14.14 Comments on the Mineral Resource Estimate

Areas of uncertainty that may materially impact the mineral resource estimate include:

- commodity price assumptions;
- pit slope angles;
- metal recovery assumptions; and
- mining and process cost assumptions.

There are no other known factors or issues that materially affect the estimate other than normal risks faced by mining projects in British Columbia in terms of environmental, permitting, taxation, socio-economic, marketing, or political factors. The author is not aware of any legal or title issues that would materially affect the mineral resource estimate.

15 MINERAL RESERVE ESTIMATES

The Baptiste Deposit is a large, near surface, bulk mineable deposit that is well suited for conventional large-scale truck and shovel operation. The Mineral Reserves for Baptiste were based on the Mineral Resource estimate described in Section 14 of this report. The Mineral Reserve estimate was performed using a metal price of US$8.75/lb for nickel. Only Davis Tube Recoverable (DTR) nickel grades were used for net smelter return (NSR). Only Measured and Indicated Mineral Resources were considered for processing, with Inferred Mineral Resources treated as waste.

This section describes the technical and economic parameters used, including but not limited to geotechnical, metallurgical, NSR, cut-off grade strategy, Lerchs Grossmann nested pit shells, and the final PFS pit design.

15.1 Geotechnical Considerations

Pit slope angles vary by sector within the open pit and are based on recommendations by Knight Piésold (Preliminary Open Pit Slope Assessment, November 1, 2021).

Pit slope designs are based on double benching of 10 m height and consider haulage ramp positioning, safety berms, and other geotechnical features required to maintain safe inter-ramp and overall slope angles.

15.2 Dilution and Mining Loss

Based on the homogeneity of the deposit and the utilization of an elevated cut-off grade (0.06% DTR Ni) during pit optimization and mine planning, it is reasonable to state that the occurrences of losses and dilution are minimal. Consequently, there was no need to incorporate provisions for ore loss or dilution in the finalized PFS mine plan.

15.3 Net Smelter Return (NSR)

To assess the value of the rock, a NSR formula was run in the block model. Only DTR nickel grades were used, and no other contributions or deleterious elements were considered for the concentrate. The inputs for the NSR calculation are detailed in Sections 15.3.1 and 15.3.1.

15.3.1 Economic Assumptions

Key economic assumptions used for mine planning are presented in Table 15-1.

Table 15-1: Key Economic Assumptions for NSR Calculations

Parameter	Value
Nickel (US$/lb)	8.75
Selling Cost (US$/lb)	0.55 (considers 95% payability and freight costs)
Net Price (US$/lb)	8.20

15.3.2 Metallurgical Recoveries

The metallurgical recovery for DTR nickel was fixed at 85% to a 63% nickel concentrate for all measured and indicated blocks with DTR nickel grade greater or equal to 0.06%.

After Whittle optimization and mine planning were finished, completion of the PFS metallurgical program and finalization of the associated recoveries were introduced, which increased the overall DTR nickel recovery to 88.7%. This consists of 82.2% recovery to flotation concentrate grading 60% nickel, and a further 6.5% recovery to a new MHP containing approximately 45% nickel. This recovery increase is deemed to have a non-material effect on the mine phases, ultimate pit design, and material above cut-off grade.

15.4 Cut-Off Grade

The mine plan includes Measured and Indicated resource categories inside the final pit design. The mine plan uses an elevated cut-off grade of 0.06% DTR nickel, which includes consideration for all site operating costs, sustaining costs, and minimum profit, representing a total of US$ 9.20/t.

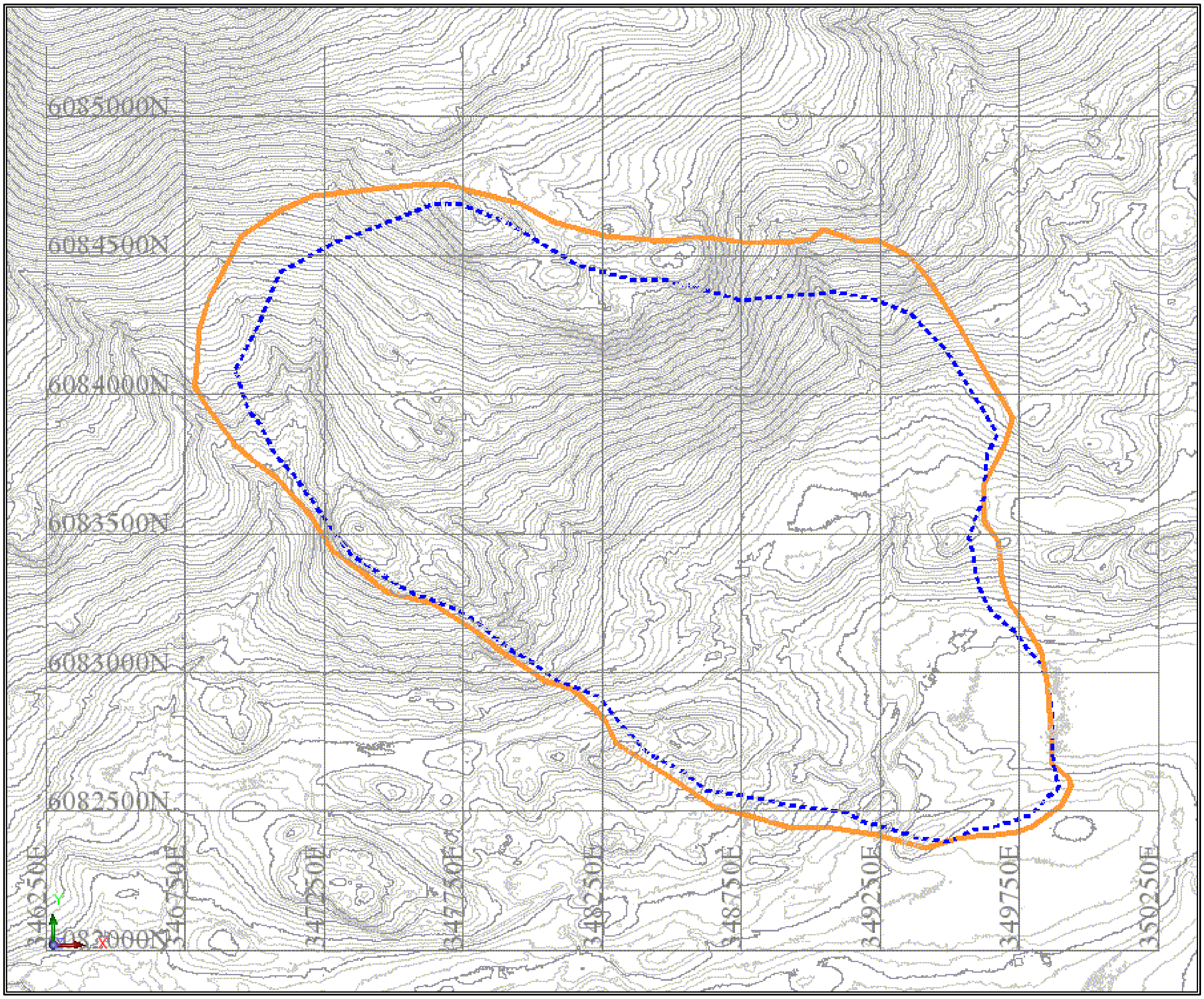
15.5 Pit Optimization

The pit optimization exercise utilizes the Lerchs-Grossman algorithm which produces a set of nested pit shells based on increasing values of the main determinant of revenue. As NSR is being used as the main revenue determinant for Baptiste, the revenue factors represent factors of the base NSR value. The resultant set of nested pit shells were then used to provide guidance for the design of the operational pit phases and the final pit selection. The pit-by-pit graphs produced and the values for the selected pit shells are presented in Figure 15-2. The Whittle run also used two “no-go” areas to protect two lakes in the vicinity of the open pit, leaving 200 m buffers between the ultimate pit rim and both Lower Baptiste Lake and Nickel Lake, respectively.

Pit 27 at a revenue factor of 0.59 was selected to guide the ultimate pit design. The selected revenue factor was due to the following considerations:

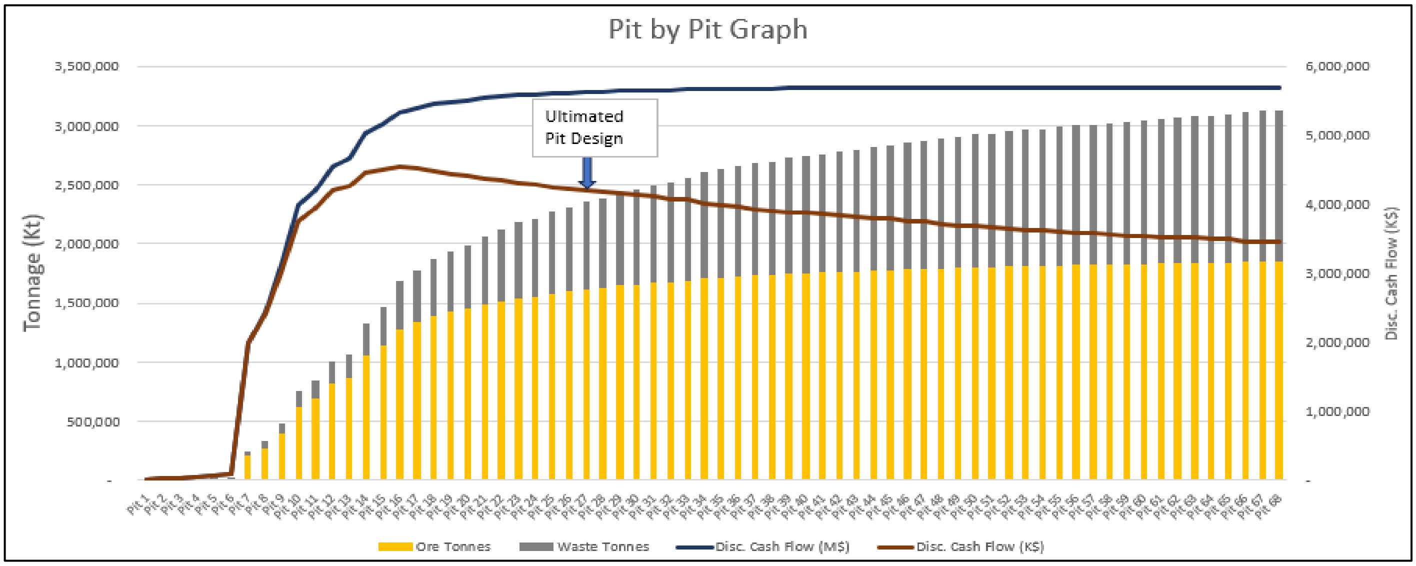
- incremental discounted cash flows are considerably reduced after this point;
- ratio between waste and ore (SR W/O) continues to increase after this point; and
- footprint availability for tailings storage within the Baptiste valley.

Figure 15-1: Pit 68 and 27 (RF 1 and 0.59 Respectively) Whittle Shell Outline



Source: TechSer, 2023.

Figure 15-2: Whittle Pit-by-Pit Graph with Selected Ultimate Pit

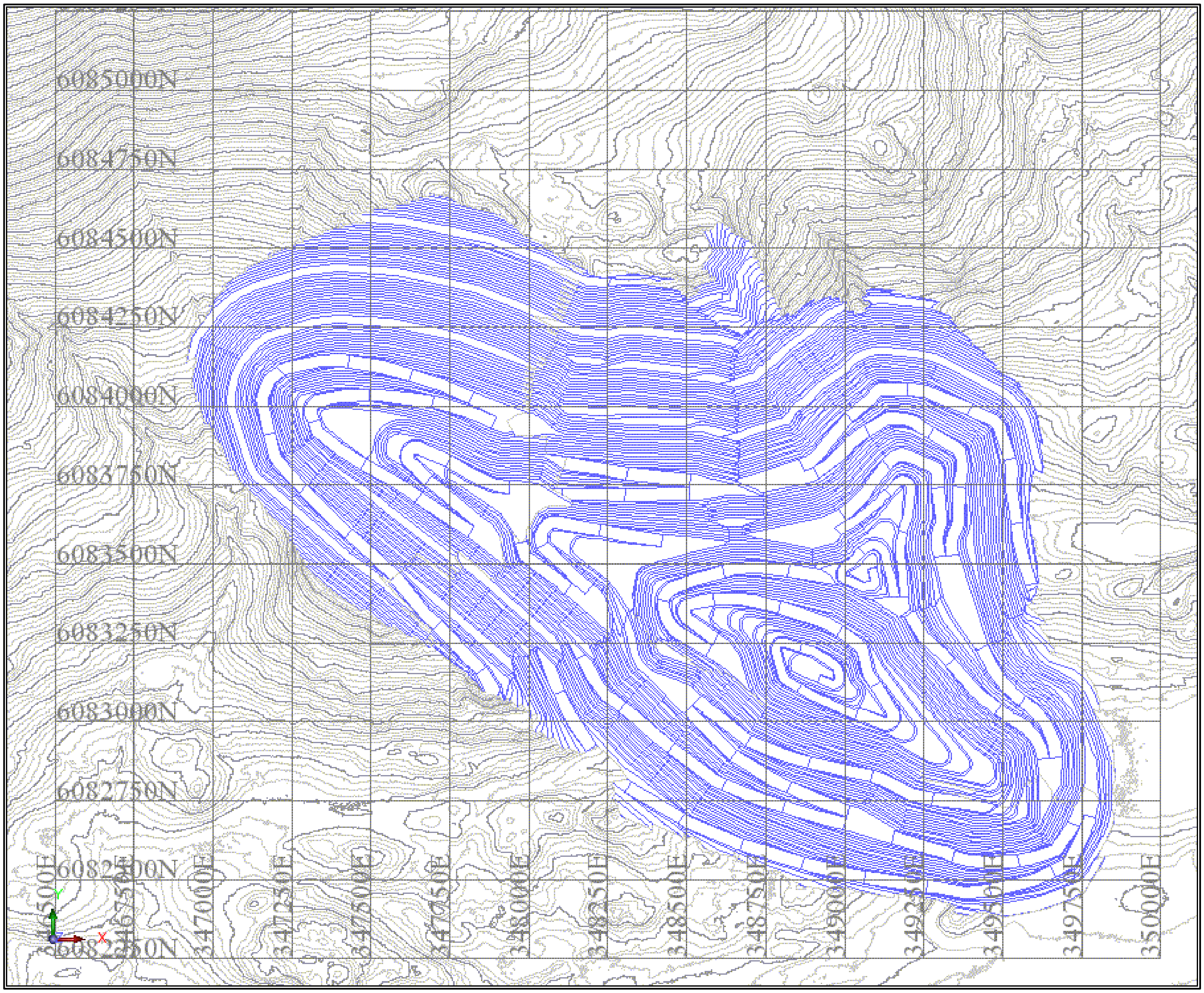


Source: TechSer, 2023.

15.6 Ultimate Pit Design

The open pit has been designed for 300 t haul trucks and 42 m^3 electric excavators. Eight mining phases have been developed to ensure the ore is release in a timely manner, to target higher-grade zones earlier in the mine life, and to reduce the initial strip ratio. The design incorporates 40 m wide haul roads with a maximum 10% grade and leaves a minimum of 120 m for operating mining width between phases.

Figure 15-3: Plan View of Baptiste Ultimate Pit Design



Source: TechSer, 2023.

15.7 Mineral Reserve Statement

The estimated Mineral Reserves are reported using a metal price of \$8.75/lb Ni and cut-off grade of 0.06% DTR Ni (around US\$9.00/t). This results in an average NSR of \$19.80/t over the planned LOM.

Mineral Reserves are reported effective September 6, 2023 using CIM Definition Standards. The qualified person for the estimate is Mr. Cristian Hernan Garcia Jimenez, P.Eng., independent consultant. Reserves are presented in Table 15-2.

Table 15-2: Baptiste Nickel Project Reserve Estimate

Category	Tonnes (kt)	Ni DTR (%)	Total Ni (%)	Contained Metal DTR Ni (kt)	Contained Metal Ni (kt)
Proven	-	-	-	-	-
Probable	1,488	0.130	0.210	1,933	3,125
Total Proven and Probable	**1,488**	**0.130**	**0.210**	**1,933**	**3,125**

Notes:

1. Mineral Reserves are reported effective September 06, 2023. The Qualified Person for the estimate is Mr. Cristian Hernan Garcia Jimenez, P.Eng, an independent consultant.
2. Mineral Reserves are reported using a fixed 0.06% Ni DTR cut-off grade, which represent approximately US$9/t NSR value, which is above the economic cut-off grade of US$5.5/t.
3. The Mineral Reserves are supported by a mine plan, based on a pit design, guided by a Lerchs Grossmann (LG) pit shell. Inputs to that process are:
   - Metal prices of Ni US$8.75/lb
   - Mining Cost US$1.98 /t moved
   - An average processing cost of US$3.72 /t milled, which includes processing and tailing storage costs.
   - General and administrative cost of US$1.10 /t milled.
   - Pit overall slope angles varying from 42 to 44 degrees.
   - A fixed process recovery of 85% for all the measured and indicated blocks above 0.06% DTR nickel grade.
4. The life-of-mine strip ratio is 0.56 (W:O), excluding capitalized pre-stripping.
5. Tonnage and contained nickel tonnes are reported in metric units and grades are reported as percentages.
6. All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.

15.8 Factors that May Affect the Mineral Reserves Estimate

Factors that may affect the Mineral Reserves estimate include changes in:

- metal prices;
- estimated Mineral Resources used to generate the mine plan;
- metallurgical factors;
- geotechnical assumptions used to determine the overall wall angles;
- capital and operating cost estimates; and
- permitting and regulatory environment in British Columbia.

There are no other known environmental, legal, title, taxation, socio-economic, marketing, political or other relevant factors that would materially affect the estimation of Mineral Reserves that are not discussed in this report.

16 MINING METHODS

The Baptiste Deposit will be mined via bulk open pit mining operations that will be developed in two phases. Phase 1 includes the first 9 years of operation (plus the preceding 2 years of mine pre-stripping) with a nominal ore processing rate of 108 kt/d. Phase 2 begins in Operating Year 10, is 19 years in duration, and coincides with a plant expansion to a nominal ore processing rate of 162 kt/d.

The Baptiste Deposit will be mined as a conventional large-scale truck and shovel operation with up to 60 Mt of material mined per annum during Phase 1, and up to 120 Mt of material mined per annum during Phase 2. The mining operation will feature 250 mm blast hole electric drills, 42 m^3 electric excavators, and 300 t class haul trucks working on nominal 10 m high benches. A flexible combination of dozers, graders, wheel loaders, and excavators will form the core of the support equipment fleet.

Costs are based on a “first principles” build-up of operating and capital costs for the life of the project with current vendor quotations for main consumables and maintenance. Capital costs for mining equipment were based on vendor budgetary quotations.

This mine sequence and phasing approach has been validated by completing a series of Whittle analyses followed by Comet runs to optimize the Project value and the in-pit backfilling opportunities.

The Mineral Resource estimate for the Project is based on the Baptiste geological block models as described in Section 14. Only Measured and Indicated Mineral Resources were used in the PFS mine plan and estimation of Mineral Reserves.

16.1 Key Design Criteria

The following mine planning design inputs were used:

- a regular resource block model on 5 m spacing in all three dimensions, with diluted grades, and resource classifications;
- topography is predominantly based on 5 m ground surface contour data based on a 2013 LiDAR-developed survey;
- inferred Mineral Resources were treated as waste;
- only DTR Ni was used as a recoverable metal in the NSR formulas (i.e. cobalt was not included in NSR calculations for the PFS Base Case); and
- Nickel Lake and Lower Baptiste Lake were kept clear of by means of a 200 m buffer zone between the respective waterbodies and the ultimate pit rim.

16.1.1 Net Smelter Revenues (NSR) Formula

The assumptions presented in Table 16-1 were used to calculate the NSR values. Only DTR Ni grades were used; no other contributions or deleterious elements were considered for the concentrate. The final NSR value was calculated as follows:

NSR = Σ$Revenue$ (Nickel) – $Selling\ Cost$ – $Royalty$

Where:

Ni $Revenue$ = DTR Ni $Grade$ (%) × DTR Ni $Recovery$ (%) × Ni $Payab$ility (%) × Ni $Price$ ($\$/lb$) × 2204.6 lb/t

$Selling\ Cost$ = Land Transport cost + $Ocean\ Freight$ + $Insurance$

$Royalty$ = % per Total revenue – concentrate transport cost

Table 16-1: Mine Design Criteria for PFS Optimization

Criteria	Value
Metal Prices	
Nickel (US$/lb)	8.75
Process Recovery	
Nickel Recovery (%)	85
Contract Terms	
Nickel Concentrate (Ni %)	63
Nickel Payability (% of LME)	95
Moisture Content (%)	3
Total Freight ($/wmt)	149.6
Others	
Royalties	
Royalty	1%

Note: The assumptions in Table 16-1 were used for mine optimization purposes and should not be confused with the final PFS criteria or results.

16.1.2 Dilution and Mining Losses

Based on the homogeneity of the Baptiste Deposit and the utilization of an elevated cut-off grade (0.06% DTR Ni) during pit optimization and mine planning, it is reasonable to state that the occurrences of losses and dilution are minimal; therefore, there was no need to incorporate provisions for ore loss or dilution in the finalized mine plan.

16.1.3 Geotechnical Considerations

Pit slope angles vary by sector within the open pit and are based on recommendations by Knight Piésold (Preliminary Open Pit Slope Assessment, November 1, 2021). The resulting pit slope designs are based on double-benching of 10 m high benches and consider haulage ramp positioning, safety berms and other geotechnical features required to maintain safe inter-ramp and overall slope angles.

Slope recommendations are summarized into three categories, namely overburden, south wall and north wall. The respective pit slope design parameters that have been used are presented in Table 16-2.

Table 16-2: Mine Design Pit Slope Recommendations

Design Sector	Max Wall Height (m)	Bench Height (m)	Bench Face Angle (°)	Bench Width (m)	Inter-ramp Angle (°)	Max. Inter-ramp Slope Height (m)	Max. Overall Slope Angle (°)
Overburden	30	10	40	8.0	27	30	
South Wall Bedrock	500	20	70	9.0	51	150	44
North Wall Bedrock	700	20	65	8.6	48	150	42

16.2 Pit Optimization

GEOVIA's Whittle™ mine optimization software was used for pit optimization. Only blocks classified as Measured and Indicated resources were considered for processing, with Inferred resources treated as waste. The main parameters in pit optimization are presented in Table 16-3. Note that these presented economic assumptions were provided by FPX.

Table 16-3: Whittle Optimization Parameters

Criteria	Value*
Metal Prices	
Nickel ($/lb)	8.75
Cobalt ($/lb)	22.5
Economic Indicators	
Diesel (US$/l)	0.90
Power (US$/kWh)	0.046
Discount rate (%)	8
Exchange Rate (C$/US$)	0.76
Mining Cost	
Mining cost (US$/t)	1.79
Incremental Cost US$/bench up	0.19
Incremental Cost US$/bench down	0.013
Process (US$/t milled)	3.72
G&A (US$/t milled)	1.11
Sustaining Capital (US$/t milled)	0.67
Total	**5.50**
Process Recovery	
Nickel Recovery (%)	85

Criteria	Value*
Contract Term	
Nickel Concentrate (Ni %)	63
Nickel Payability (% of LME)	95
Moisture Content	3
Total Freight ($/wmt)	149.6
Others	
Royalties	
Royalty (%)	1

* Note: The assumptions in Table 16-3 were used for pit optimization purposes and should not be confused with the final PFS criteria or results.

The pit optimization exercise utilized a Lerchs-Grossman algorithm to produce a set of nested pit shells based on increasing values of the main determinant of revenue. As NSR is being used for Baptiste, the revenue factors represent factors of the base NSR value. The resultant set of nested pit shells were then used to provide guidance for the design of the operational pit phases and the final pit selection. The pit-by-pit graphs produced and the values for the selected pit shells are presented in Figure 16-4. The Whittle run also used two "no-go" areas avoiding impact to Nickel and Lower Baptiste Lakes via a 200 m buffer.

16.2.1 Pit Limits

The result of pit optimization runs are presented in Table 16-4 and Figure 16-1. Pit shell 27 at a revenue factor of 0.59 was selected to guide the ultimate pit design. The selected revenue factor was due to the following considerations:

- incremental discounted cash flows are considerably reduced after this point;
- ratio between waste and ore (SR W/O) continues to increase after this point; and
- footprint available for tailings storage within the Baptiste valley.

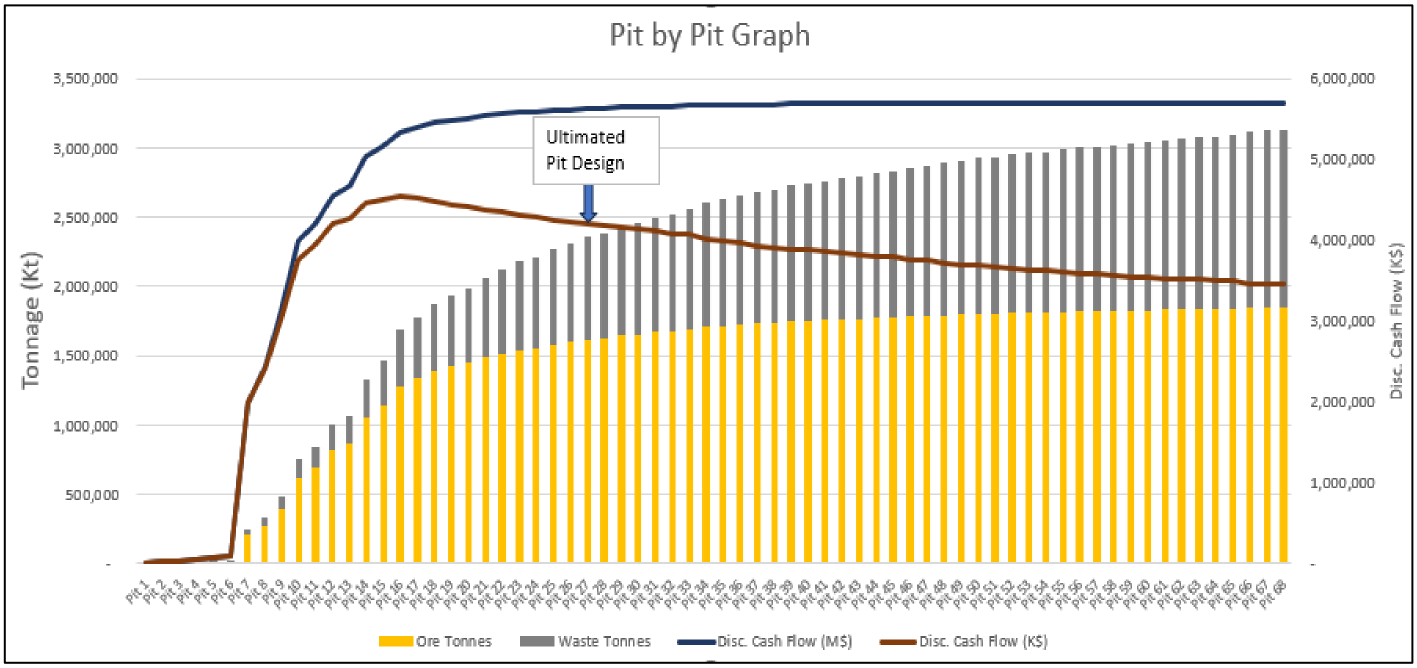
Table 16-4: Pit Optimization Run Results

Pit #	Revenue Factor	Total (Mt)	Waste (Mt)	Ore (Mt)	Discounted Cash Flow (US$ M)	Strip Ratio (W:O)
Pit 1	0.33	1	0	1	16	0.07
Pit 2	0.34	2	0	1	21	0.07
Pit 3	0.35	2	0	2	31	0.06
Pit 4	0.36	3	0	3	45	0.06
Pit 5	0.37	5	0	4	60	0.05
Pit 6	0.38	7	0	7	92	0.05
Pit 7	0.39	251	44	207	1,979	0.22
Pit 8	0.40	332	57	275	2,436	0.21
Pit 9	0.41	486	84	401	3,109	0.21
Pit 10	0.42	751	129	623	3,982	0.21
Pit 11	0.43	843	148	695	4,212	0.21
Pit 12	0.44	1,005	183	822	4,553	0.22
Pit 13	0.45	1,070	198	872	4,669	0.23
Pit 14	0.46	1,331	272	1,058	5,025	0.26
Pit 15	0.47	1,466	329	1,136	5,154	0.29
Pit 16	0.48	1,683	400	1,283	5,337	0.31
Pit 17	0.49	1,776	441	1,335	5,397	0.33
Pit 18	0.50	1,875	486	1,390	5,453	0.35
Pit 19	0.51	1,938	515	1,422	5,484	0.36
Pit 20	0.52	1,989	541	1,449	5,508	0.37
Pit 21	0.53	2,062	579	1,483	5,537	0.39
Pit 22	0.54	2,118	608	1,510	5,558	0.40
Pit 23	0.55	2,181	641	1,540	5,580	0.42
Pit 24	0.56	2,214	660	1,554	5,590	0.42
Pit 25	0.57	2,270	692	1,578	5,606	0.44
Pit 26	0.58	2,315	717	1,597	5,617	0.45
Pit 27	0.59	2,361	744	1,617	5,629	0.46
Pit 28	0.60	2,385	759	1,627	5,634	0.47
Pit 29	0.61	2,433	787	1,646	5,644	0.48
Pit 30	0.62	2,457	802	1,655	5,648	0.48
Pit 31	0.63	2,496	827	1,669	5,655	0.50
Pit 32	0.64	2,526	846	1,680	5,660	0.50
Pit 33	0.65	2,552	862	1,689	5,664	0.51
Pit 34	0.66	2,609	899	1,709	5,672	0.53
Pit 35	0.67	2,633	916	1,717	5,675	0.53

Pit #	Revenue Factor	Total (Mt)	Waste (Mt)	Ore (Mt)	Discounted Cash Flow (US$ M)	Strip Ratio (W:O)
Pit 36	0.68	2,651	928	1,723	5,677	0.54
Pit 37	0.69	2,685	950	1,735	5,681	0.55
Pit 38	0.70	2,700	961	1,739	5,683	0.55
Pit 39	0.71	2,726	978	1,748	5,686	0.56
Pit 40	0.72	2,742	990	1,752	5,687	0.56
Pit 41	0.73	2,762	1,003	1,759	5,689	0.57
Pit 42	0.74	2,781	1,017	1,764	5,690	0.58
Pit 43	0.75	2,791	1,024	1,767	5,691	0.58
Pit 44	0.76	2,820	1,045	1,774	5,693	0.59
Pit 45	0.77	2,826	1,050	1,776	5,693	0.59
Pit 46	0.78	2,860	1,074	1,786	5,695	0.60
Pit 47	0.79	2,865	1,077	1,788	5,696	0.60
Pit 48	0.80	2,888	1,095	1,793	5,697	0.61
Pit 49	0.81	2,908	1,109	1,799	5,698	0.62
Pit 50	0.82	2,927	1,124	1,803	5,698	0.62
Pit 51	0.83	2,933	1,128	1,805	5,699	0.62
Pit 52	0.84	2,955	1,145	1,810	5,700	0.63
Pit 53	0.85	2,964	1,152	1,812	5,700	0.64
Pit 54	0.86	2,966	1,153	1,813	5,700	0.64
Pit 55	0.87	2,988	1,170	1,818	5,700	0.64
Pit 56	0.88	3,001	1,180	1,821	5,701	0.65
Pit 57	0.89	3,007	1,185	1,822	5,701	0.65
Pit 58	0.90	3,017	1,193	1,824	5,701	0.65
Pit 59	0.91	3,034	1,205	1,829	5,701	0.66
Pit 60	0.92	3,039	1,210	1,830	5,701	0.66
Pit 61	0.93	3,060	1,226	1,834	5,702	0.67
Pit 62	0.94	3,068	1,232	1,836	5,702	0.67
Pit 63	0.95	3,074	1,237	1,837	5,702	0.67
Pit 64	0.96	3,080	1,242	1,838	5,702	0.68
Pit 65	0.97	3,090	1,250	1,840	5,702	0.68
Pit 66	0.98	3,114	1,269	1,845	5,702	0.69
Pit 67	0.99	3,128	1,280	1,848	5,702	0.69
Pit 68	1.00	3,128	1,280	1,848	5,702	0.69

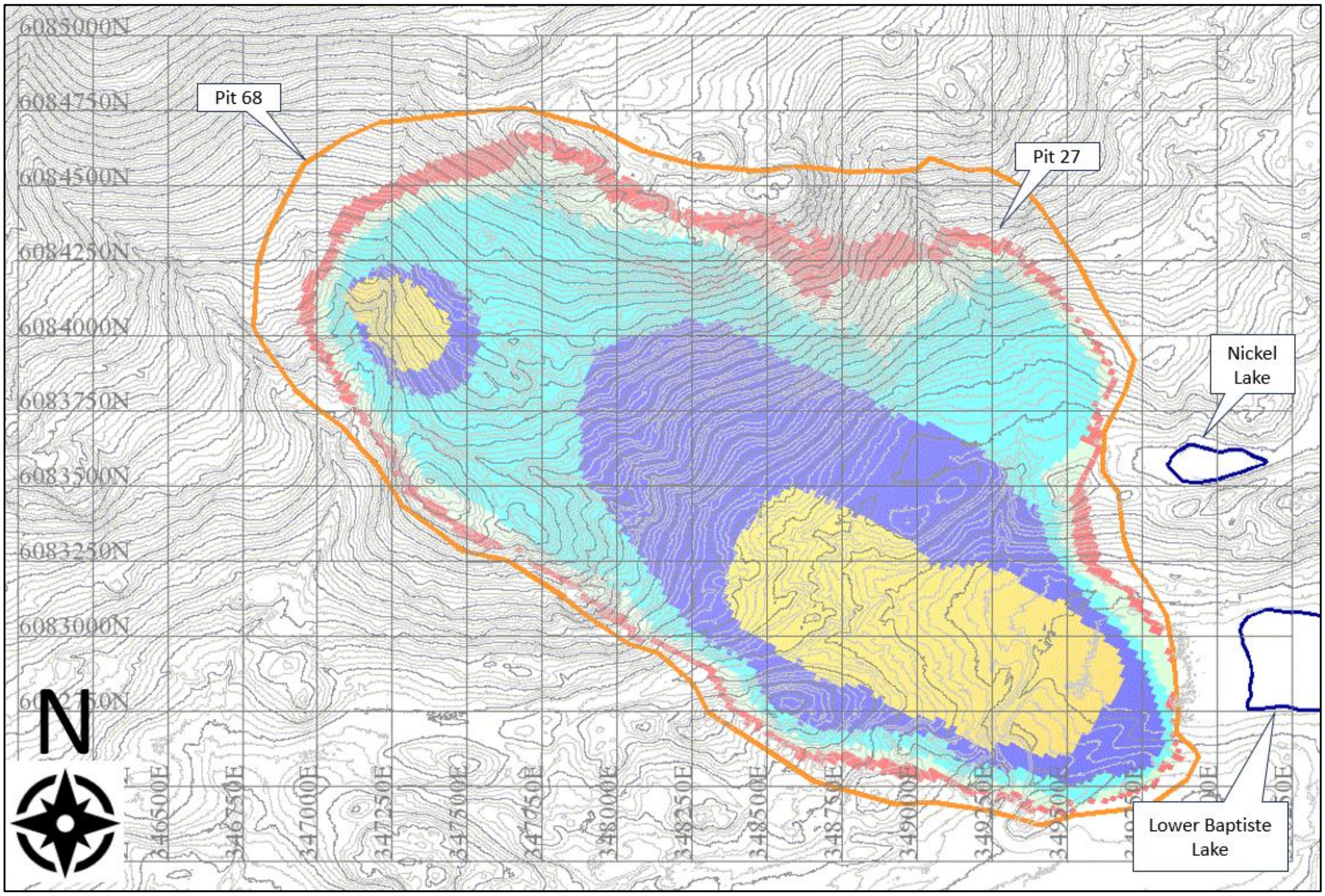
The highlighted pits represent the Whittle pit shells that best align with the detailed pit phase designs. Also included are the results for Pit 68 (outline), which represents the pit shell using a revenue factor of 1.00. Pit 68 represents the economic limit of expansion based on the stated set of assumptions. The triangulation of the various pits shells on surface are presented in Figure 16-2, shown with 5 m topography contours lines.

Figure 16-1: Pit-by-Pit Run Results



Source: TechSer, 2023.

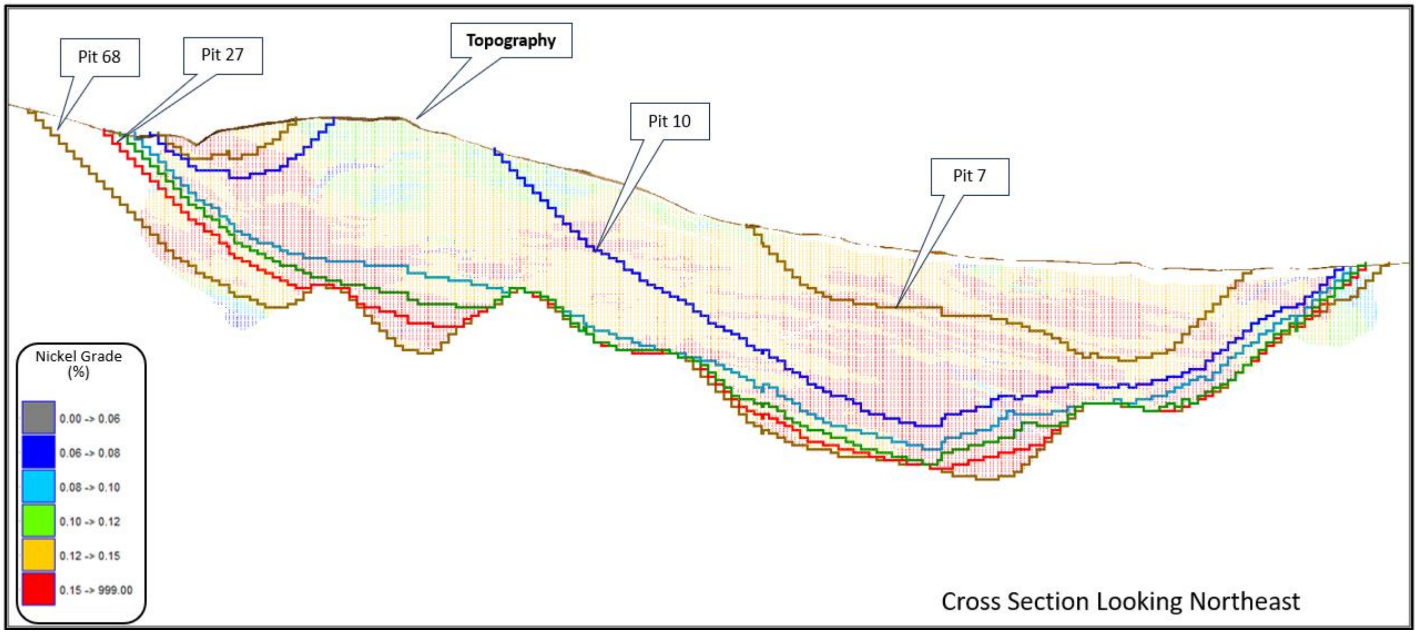
Figure 16-2: Pit Shells in Plan View



Source: TechSer, 2023.

Figure 16-3 presents a long section displaying Measured and Indicated blocks above the 0.06% DTR nickel cut-off grade, along with the selected shells that guide the phase design. The figure also highlights the distribution of higher-grade material within the deposit.

Figure 16-3: Pit Shells in Section View



Source: TechSer, 2023.

16.2.2 Sensitivities

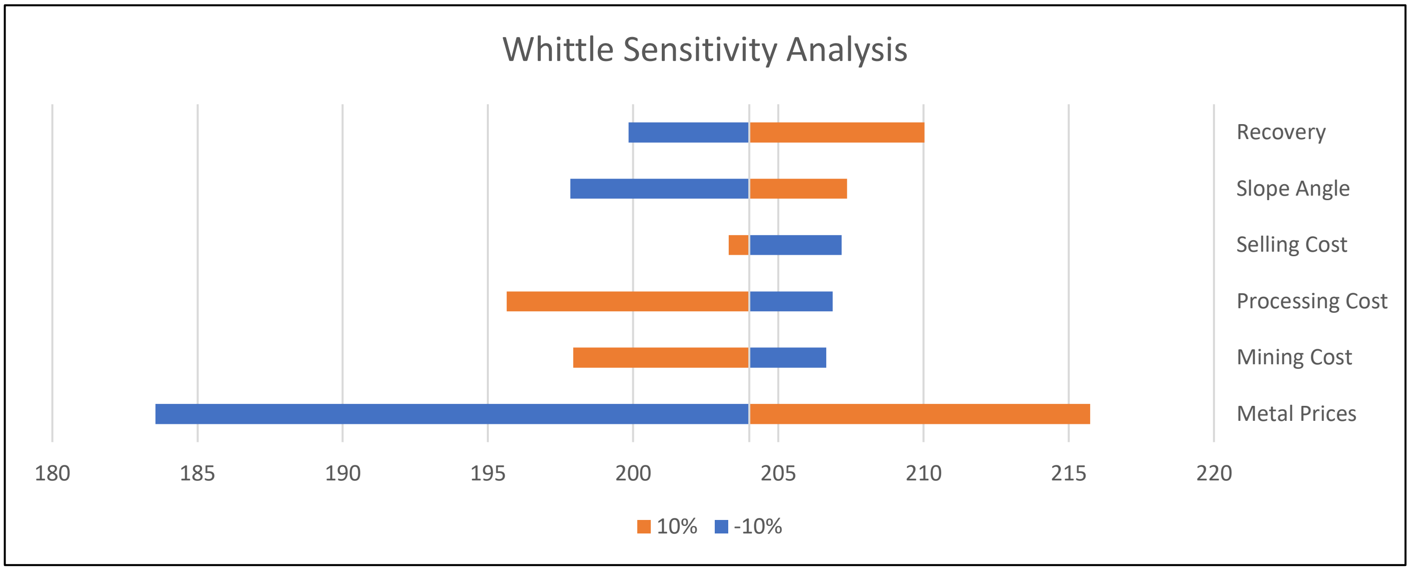
Figure 16-4 presents the results of various sensitivity analyses conducted in Whittle which assess the potential impact on production, cash flow, LOM, and other key project drivers. These sensitivities alter factors such as metal price, slope angles, mining costs, processing costs, and process recovery, as presented in Table 16-5.

Table 16-5: Pit Optimization Sensitivity Criteria

Item	Unit	Bottom Factor	Top Factor
Metal Prices	%	-10	+10
Mining Cost	%	-10	+10
Processing Cost	%	-10	+10
Selling Cost	%	-10	+10
Slope Angle	degrees	-2d	+2d
Metallurgical Recoveries*	%	-3	+3

* Metallurgical recoveries are absolute percentage points (not relative).

Figure 16-4: Pit Optimization Sensitivity Results



Source: TechSer, 2023.

Figure 16-4 provides a clear illustration of the relative impact of key project factors. It indicates that changes in metal prices have the most pronounced effect, followed by metallurgical recoveries and processing costs. For instance, a 10% increase in metal prices has a positive impact of 6% on metal content, while a 10% decrease in metal price has a 9.9% decrease in metal content.

16.3 Mine Design

The detailed PFS mine design considers typical parameters for a conventional large-scale truck and shovel operation, as well as the geotechnical recommendations provided by Knight Piésold. Key mine design criteria established for the Project are presented in Table 16-6.

Table 16-6: Key Mine Design Criteria for PFS Mine Design

Parameters	Value
Ramp Width (m)	40
Maximum ramp grade (%)	10
Minimum mining width (m)	120
Bench face angle (degrees)	Varies between 70° to 65° by design sector
Inter-ramp angle (degrees)	Varies between 42° to 44° by design sector
Bench height (m)	10 (double bench 20m)
Berm width (m)	Varies between 8.6 to 9 by design sector
Inter-ramp slope height (m)	150
Geotechnical berm width (m)	30

The Baptiste mine phase sequence was mainly based on the Whittle shell sequence. Four different scenarios were evaluated in order to determine the best combination of pit economics and maximizing in-pit tailings deposition and waste backfilling opportunities. This first stage of design was done at a high level to ensure the overall slope angles were followed and the minimum operational areas were respected. These 4 high level scenarios are described below:

- Design Scenario 1: Honour the whittle shells (without modification).
- Design Scenario 2: Accelerate the southern extent of pit phase to allow for earlier in-pit tailings deposition.
- Design Scenario 3: Modify the north extent of pit phase to allow for additional in-pit tailings deposition (in addition to what is plausible under design scenario 1).
- Design Scenario 4: Removes the final phase (phase 8) from Design Scenario 2.

In all cases, the same initial 3 phases were adopted such that some of the highest-grade ore in the Baptiste Deposit was mined in the first 10 years of production. For this reason, all 4 scenarios have similar discounted cashflows. Phases 4, 5, 6, 7 and 8 were changed in order to maximize in-pit tailings storage deposition. A summary table of a relative comparison of discounted cashflow is presented in Table 16-7.

Table 16-7: High level Mine Design Scenarios for Baptiste PFS

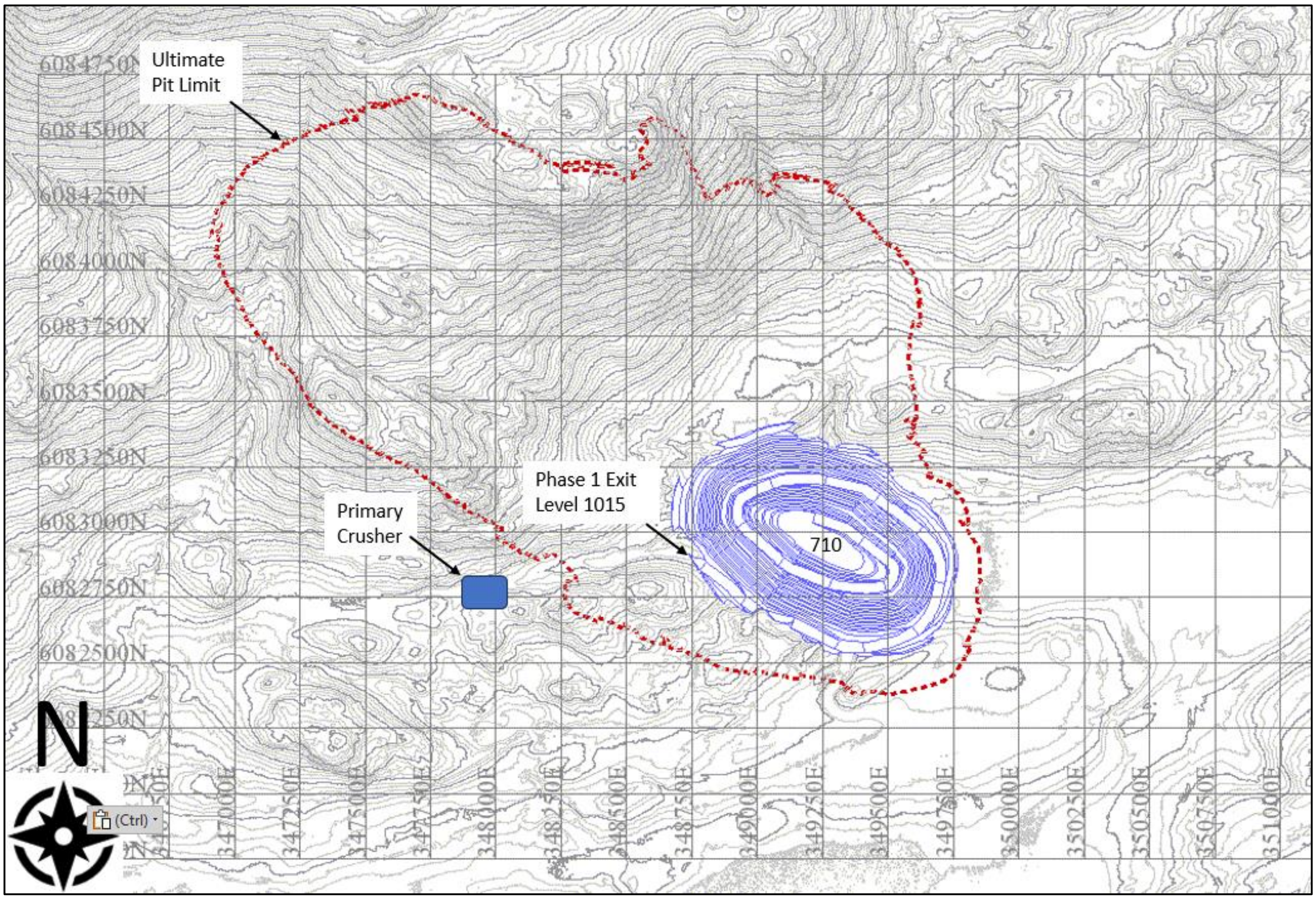
	Design 1	Design 2	Design 3	Design 4
Pre Tax Disc. Cashflow (compared to Base Case)	0	+2%	+0.6%	-4.3%
Description of the Case	Honour the Whittle shells	Accelerate the southern “phase” to allow in-pit storage	Modify the northern “phase” to allow extra in-pit storage	Takes scenario 2 and eliminates the last phase.

After this exercise it was decided to continue with design scenario 3 for the PFS. Scenario 3 allows approximately 70 Mm^3 of in-pit tailings storage after Year 21, and an additional 60 Mm^3 of storage after Year 25 with minimal impact to economics when compared to design scenario 2. Once the final scenario was selected, mine phases were completed at a PFS level.

16.3.1 Mine Phase 1 Design

Mine Phase 1 for Baptiste is the starter pit of the Project. No final pit walls are achieved in this first mining phase. Mining begins at elevation 980 masl and continues to a bottom elevation of 710 masl. This initial phase contains approximately 175 Mt of rock at a 0.32 strip ratio (W:O). Approximately 130 Mt of material is ore at an average grade of 0.142% DTR Ni. The starter pit location targets some of the highest-grade ore in the Baptiste Deposit. The waste from this mining phase will be primarily hauled to the tailings facility starter dams located to the south of the pit. The Mine Phase 1 design is presented in Figure 16-5.

Figure 16-5: Mine Phase 1 Plan

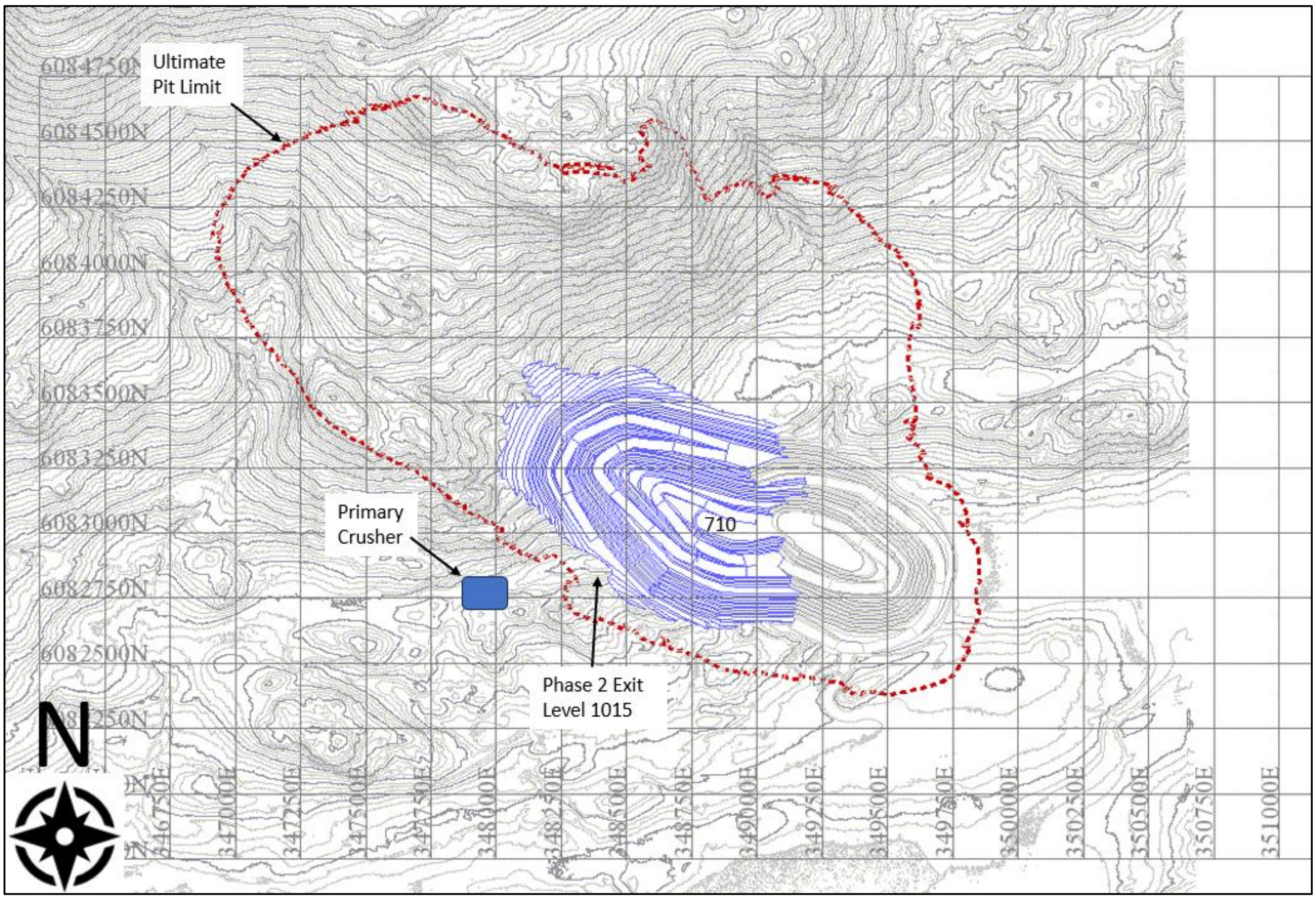


Source: TechSer, 2023.

16.3.2 Mine Phase 2 Design

Mine Phase 2 will be a pushback of Mine Phase 1 to the northwest. Similarly to Mine Phase 1, no final walls are achieved in Mine Phase 2. This mining phase will maintain the existing pit bottom at elevation 710 masl. A total of 200 Mt will be excavated in Mine Phase 2 with a 0.53 strip ratio (W:O). Approximately 132 Mt is ore at an average grade of 0.137% DTR Ni. Some Mine Phase 2 stripping begins in Year -2 in order to provide sufficient waste for tailings facility starter dam construction during Years -2 and -1. The Mine Phase 2 design is presented in Figure 16-6.

Figure 16-6: Mine Phase 2 Plan

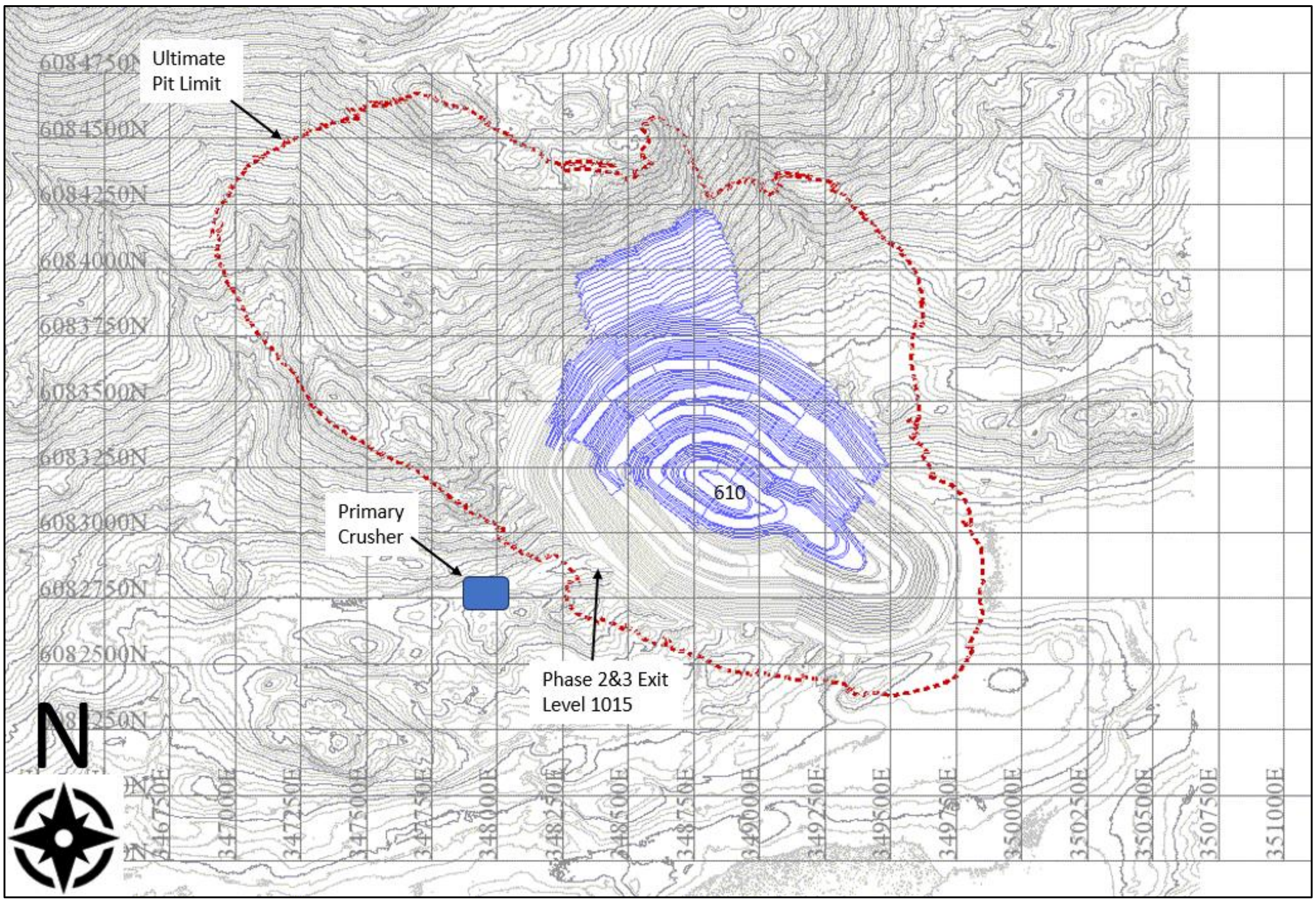


Source: TechSer, 2023.

16.3.3 Mine Phase 3 Design

Mine Phase 3 contains approximately 220 Mt of rock with a 0.28:1 strip ratio (W:O). This phase includes a pushback to the north of the pit. This phase contains approximately 170 Mt of ore at an average grade of 0.133% DTR Ni. The waste from the upper benches is hauled through external ramps on the eastern side of the pit. This phase will deepen the pit by 10 benches to the elevation 610 masl. No final pit walls are achieved during the excavation of this mining phase. The Mine Phase 3 design is presented in Figure 16-7.

Figure 16-7: Mine Phase 3 Plan

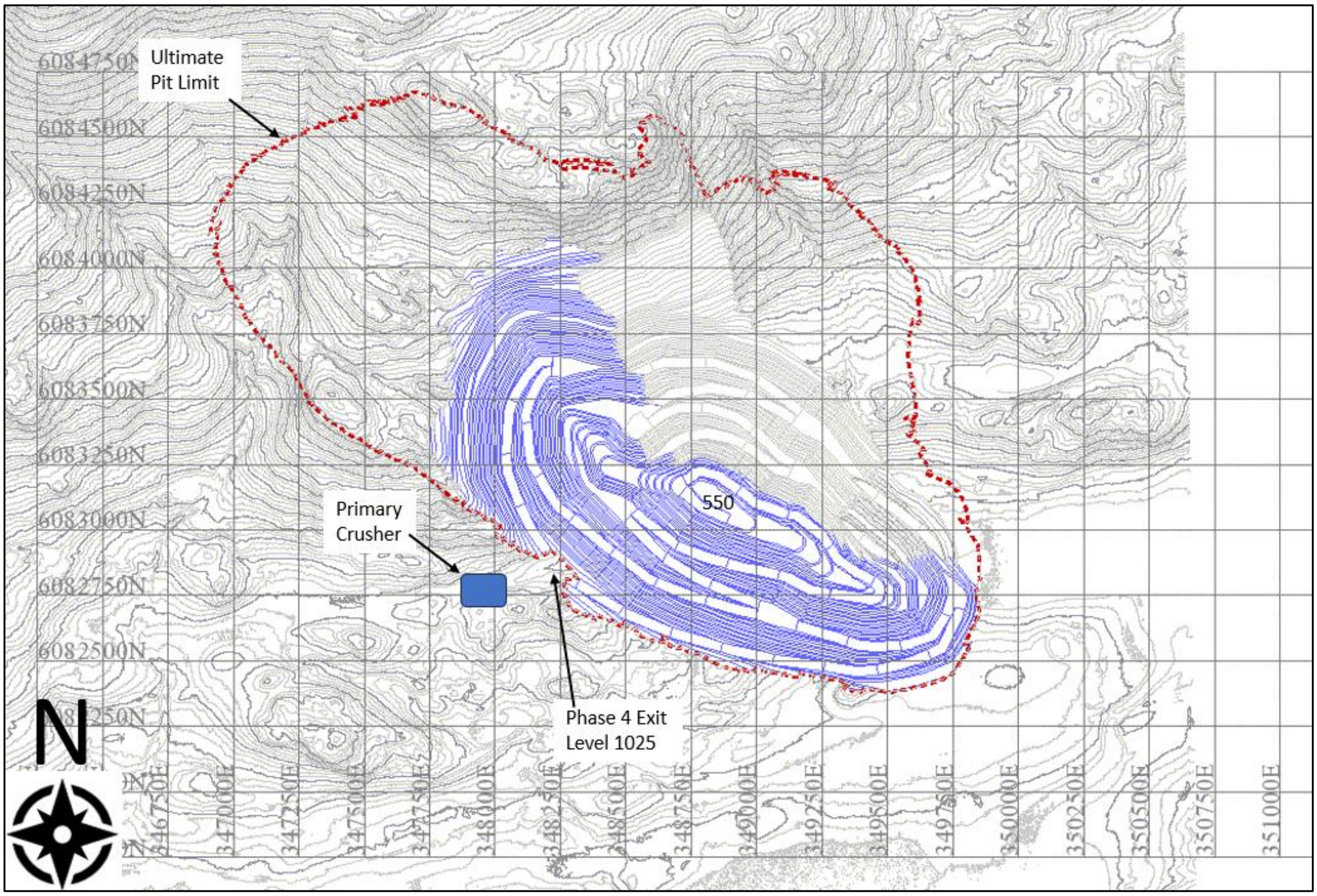


Source: TechSer, 2023.

16.3.4 Mine Phase 4 Design

Mine Phase 4 contains approximately 390 Mt of rock with a 0.77 strip ratio (W:O). This phase is a pushback to the south, it is the first phase to achieve a final pit wall. This phase contains approximately 220 Mt of ore at an average grade of 0.135% DTR Ni. The ramp from the pit bottom at 550 masl daylights to the south at the 1,025 masl elevation. The Mine Phase 4 design is presented in Figure 16-8.

Figure 16-8: Mine Phase 4 Plan

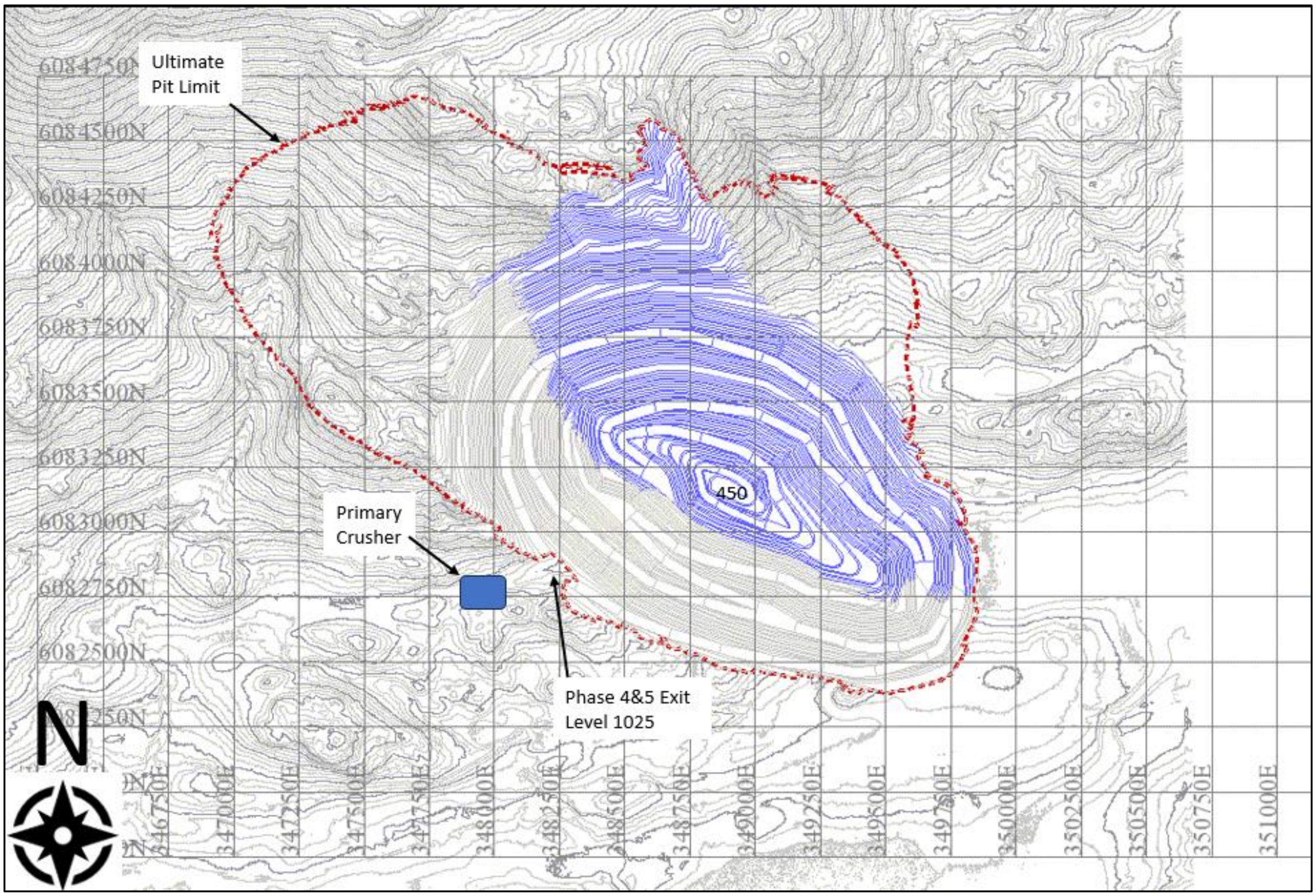


Source: TechSer, 2023.

16.3.5 Mine Phase 5 Design

Mine Phase 5 contains approximately 350 Mt of rock with a 0.56 strip ratio (W:O). Approximately 225 Mt is ore at an average grade of 0.131% DTR Ni. This phase is a pushback to the northeast. This phase achieves the final pit to the east and achieves the bottom of the southeastern pit at elevation 450 masl. The upper benches will use temporary ramps in topography, while the bottom benches will connect with the northwest ramp of Mine Phase 4. This phase is completed in Year 21, at which point backfilling waste or tailings is possible up to elevation 730 masl. The Mine Phase 5 design is shown in Figure 16-9 below.

Figure 16-9: Mine Phase 5 Plan

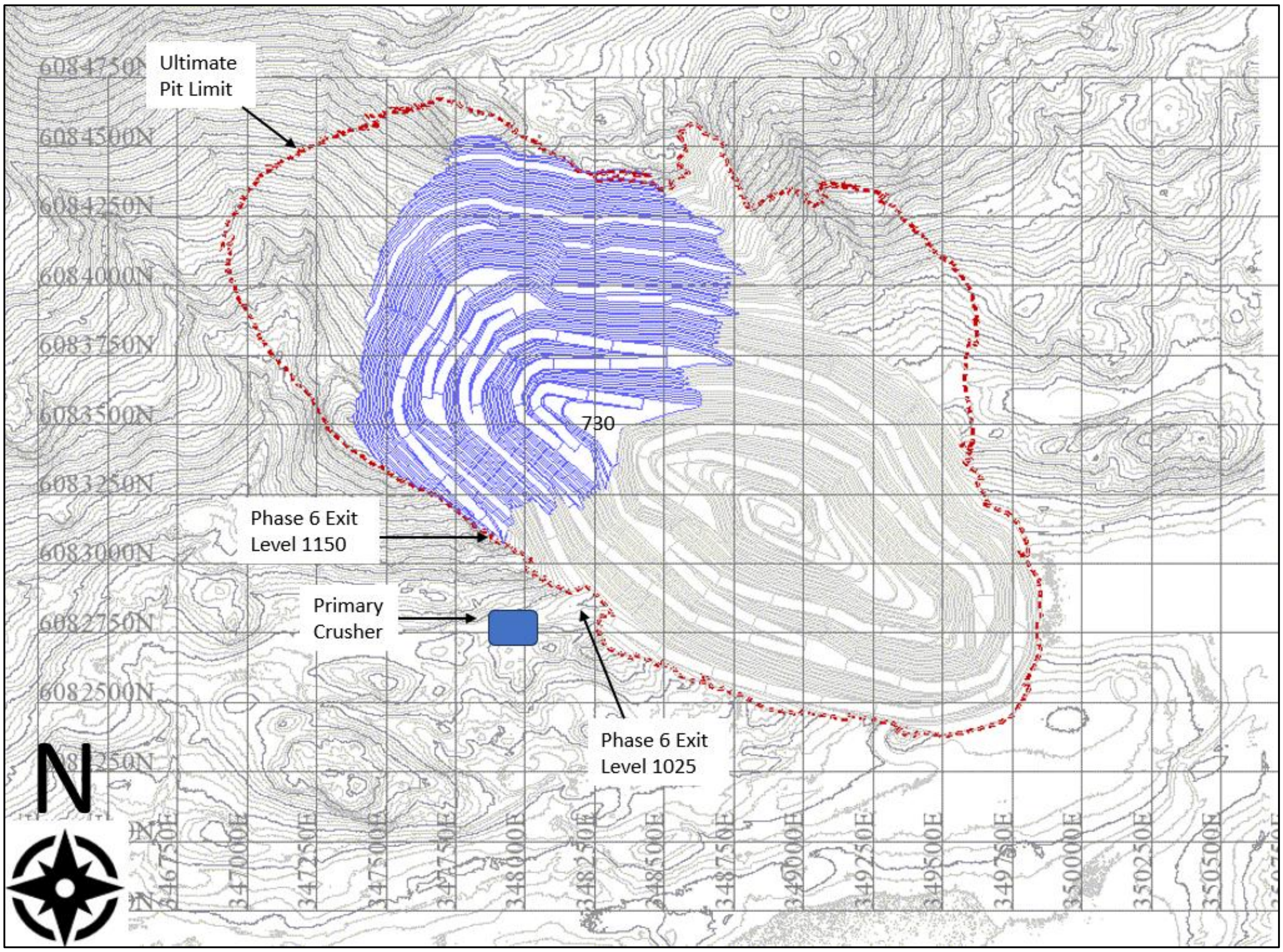


Source: TechSer, 2023.

16.3.6 Mine Phase 6 Design

Mine Phase 6 contains approximately 470 Mt of rock with a 0.82 strip ratio (W:O). Approximately 257 Mt is ore with an average grade of 0.120% DTR Ni. This phase is a pushback to the northwest and achieves final pit in two small areas of the northeast and southwest pit wall. The upper benches will use temporary ramps in topography while the bottom benches will use a combination of temporary ramps to connect with the northwest ramp of Mine Phase 4 and a ramp that exits at elevation 1150 masl. This phase is depleted in Year 25, at which point backfilling waste rock or tailings is possible up to elevation 730 masl. The Mine Phase 6 design is presented in Figure 16-10.

Figure 16-10: Mine Phase 6 Plan

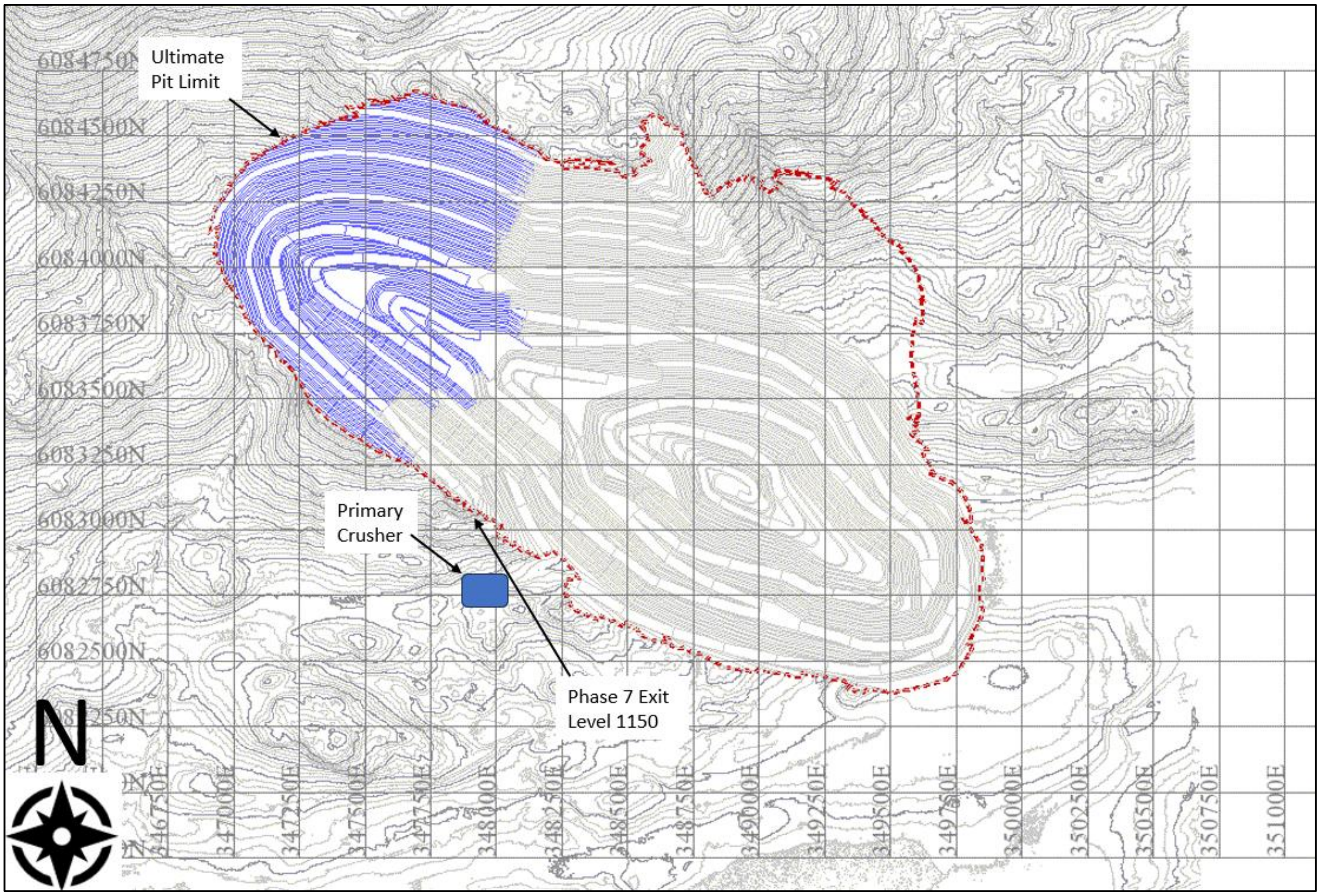


Source: TechSer, 2023.

16.3.7 Mine Phase 7 Design

Mine Phase 7 contains approximately 373 Mt of rock with a 0.60 strip ratio (W:O). Approximately 233 Mt is ore with an average grade of 0.129% DTR Ni. This phase is a pushback to the northwest and achieves the final pit all around the northwest area. Most of the waste will be hauled through temporary ramps in topography. Mine Phase 7 reaches a bottom elevation of 810 masl. The Mine Phase 7 design is presented in Figure 16-11.

Figure 16-11: Mine Phase 7 Plan

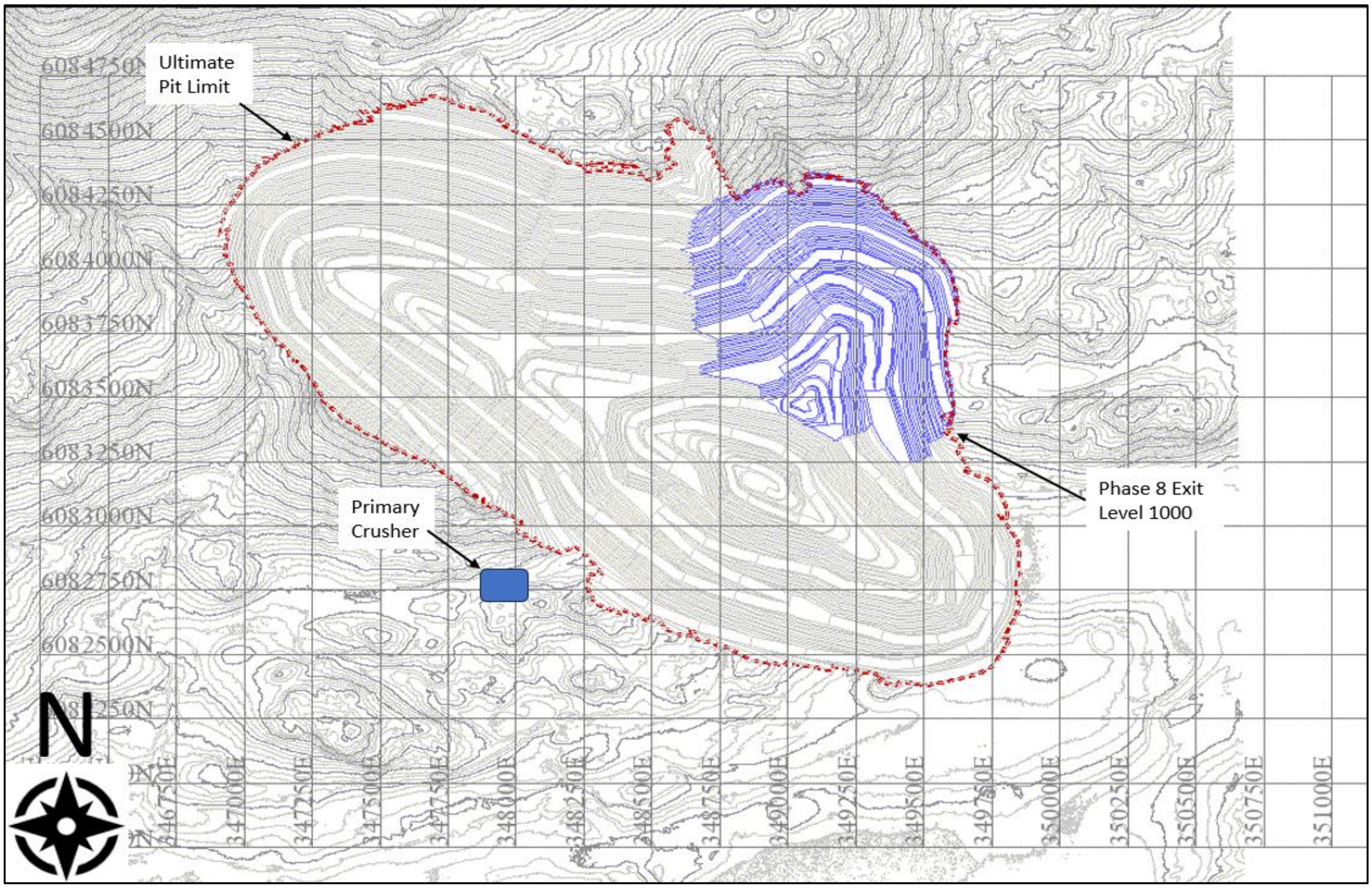


Source: TechSer, 2023.

16.3.8 Mine Phase 8 Design

Mine Phase 8 is the final mining phase of the mine design and pushes the northeast wall of the pit back to the ultimate optimized pit. This phase contains approximately 190 Mt of rock with a 0.55 strip ratio (W:O). Approximately 123 Mt is ore with an average grade of 0.117% DTR Ni. Most of the waste from this phase will be hauled through the southeastern boundary of the pit and some is placed at the bottom of Mine Phase 5 as backfill. Mine Phase 8 reaches a bottom elevation of 650 masl. The Mine Phase 8 design is presented in Figure 16-12.

Figure 16-12: Mine Phase 8 Plan



Source: TechSer, 2023.

16.3.9 Pit Summary

Table 16-8 presents tonnages and grades for all material within the eight mining phases designed for Baptiste. The data is based on the selected cut-off for the project (0.06% DTR Ni). Note that in the optimization and design, all Inferred material has been considered as waste. With further drilling, there is the potential for upgrading this material to Indicated or Measured status, which could contribute an additional 44 Mt of ore to the project.

Table 16-8: Summary of Mine Phase Tonnages and Grades

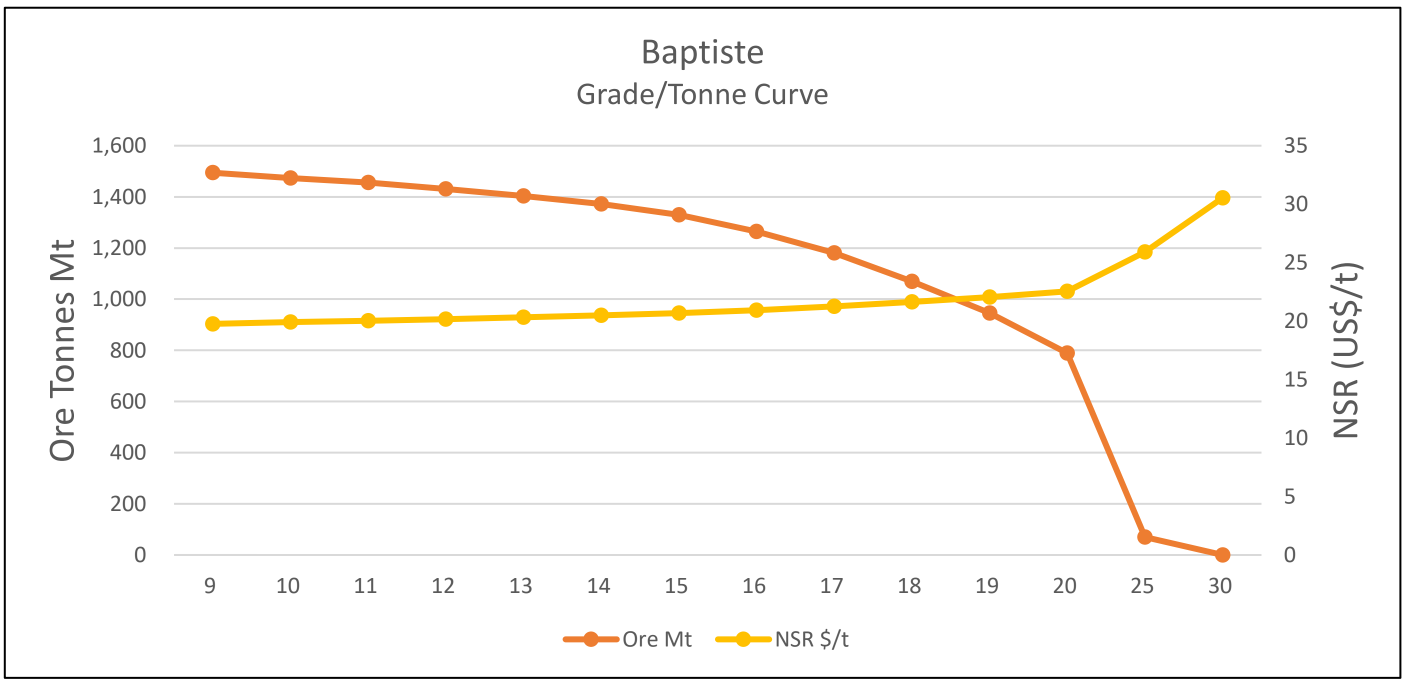
Phase	Ore (kt)	DTR Ni (%)	DTR Co (%)	DTR Fe (%)	NSR ($/t)	Waste (kt)	Inferred Material (kt)	Total Material (kt)
Phase 1	131,485	0.142	0.004	2.8	21.6	42,888	134	174,507
Phase 2	132,461	0.137	0.004	2.7	20.8	70,827	0	203,287
Phase 3	170,050	0.133	0.004	2.5	20.2	43,876	3,156	217,081
Phase 4	221,215	0.135	0.004	2.6	20.5	169,365	1,847	392,428
Phase 5	225,899	0.131	0.003	2.5	19.9	113,860	12,478	352,237
Phase 6	256,899	0.120	0.003	1.9	18.3	205,239	6,228	468,365
Phase 7	233,329	0.129	0.003	2.0	19.6	127,972	12,125	373,426
Phase 8	116,384	0.117	0.003	2.5	17.8	57,943	8,254	182,581
Total	**1,487,721**	**0.130**	**0.004**	**2.39**	**19.77**	**831,970**	**44,221**	**2,363,912**

Table 16-9 presents ore tonnages within the ultimate pit design at varying NSR cut-off grades (not including Inferred material). From this information we can conclude that the Baptiste operation is not very sensitive to changes in cut-off grade between US$9/t and US$13/t due to the homogeneity of the deposit. Note that the economic cut-off is approximately US$5.5/t. This relationship can be clearly seen in the steep ore tonnage line of the grade-tonnage curve presented in Figure 16-13.

Table 16-9: Baptiste Ore Tonnage and Grade at Varying NSR Cut-off Grades

Cut-off NSR ($US)	Ore (Mt)	DTR Ni (%)	NSR ($/t)
9.00	1,494	0.130	19.77
10.00	1,474	0.131	19.91
11.00	1,455	0.132	20.03
12.00	1,430	0.133	20.18
13.00	1,403	0.134	20.32
14.00	1,371	0.135	20.48
15.00	1,329	0.136	20.67
16.00	1,264	0.138	20.93
17.00	1,180	0.140	21.25
18.00	1,069	0.142	21.64
19.00	945	0.145	22.05
20.00	789	0.148	22.54
25.00	69	0.170	25.90
30.00	0	0.201	30.54

Figure 16-13: Baptiste Grade-Tonnage Summary of Measured and Indicated Reserves



Source: TechSer, 2023.

16.4 Mine Waste Management

Waste materials produced from mining activities include:

- ore to be temporarily stockpiled;
- Class A and Class B waste rock to be used for the tailings facility dam and causeway construction or deposited within the open pit in later mine stages (Years 23 to 26); and
- overburden materials from stripping the open pit area, which will be managed in a stockpile prior to use as a reclamation material in closure.

Mine waste movement is further described in the following subsections.

16.4.1 Operational Ore Stockpile

The operational ore stockpile has been designed to manage up to 30 Mt of rock. The stockpile includes berm with widths up to 100 m to provide access to the adjacent overburden stockpile. The operational ore stockpile will serve to stockpile ore generated during pre-stripping and lower-grade ore from the initial operating years. Stockpiled ore will be reclaimed to supplement open pit production maintaining the feed to the concentrator such that it is fully utilized. Any remaining material will be fed through the concentrator at the end of mine life, meaning the operational ore stockpile will be completely removed upon project completion.

16.4.2 Tailings Facility Dams and Internal Causeway

The tailings facility includes a North Dam, West Dam, and South Dam. The dams are all located to the south of the open pit and will be constructed using overburden and suitable waste rock material from the pit. The material will be sourced largely from Mine Phase 1 and 2. There is additional opportunity for waste rock management within a causeway to be constructed through the centre of the tailings facility serving a secondary function as an access corridor for the reclaim water return pipelines that will recover water for process facility demands. More information about the tailings facility geometry and construction methods are described in Section 18.7 of this report.

16.4.3 Overburden Stockpile

The overburden stockpile is designed to manage up to 50 Mt of overburden material. Most of the suitable overburden mined in pre-stripping and the early operating years will be used in the tailings facility dam construction. The stockpile has been designed with a dump face angle is 37° (angle of repose) with 30 m safety benches placed at 25 m vertical intervals. This results in an overall slope angle of approximately 20°, which can be further optimized in the next stage of study. The overburden material will be used for tailings facility reclamation, meaning the facility will be completely removed upon project completion. Overburden design is presented in Figure 16-14.

16.4.4 In-Pit Waste Backfill

The PFS mining sequence allows for the start of backfilling waste into the bottom of the Mine Phase 5 pit (Year 22 of operations). However, given the proximity and continued ease of waste transport to the tailings facility, in-pit waste backfill occurs in Years 23-26. In-pit tailings deposition will commence thereafter, from Year 27 to end of operations.

Table 16-10 presents a summary of the approximate total design capacity for each waste storage and the final material placement.

Table 16-10: Storage Capacity by Waste and Ore Storage Facility

Facility	Unit	Total Capacity
Overburden stockpile	kt	50,000
Tailings facility dams	kt	675,000
Tailings facility causeway	kt	120,000
In-Pit waste rock backfill	kt	100,000
Operational ore stockpile	kt	30,000

16.5 Production Schedule

The mine plan presented in this report was developed using Comet, a mining schedule optimization software from Strategy Optimization Solutions Pty Ltd. The phase design previously described was used as input for Comet, as well as the following parameters:

- block model, metallurgical recoveries and NSR;
- PFS economic parameters;
- high level cycle times;
- descent rates limited to 10 benches per annum (bench productivities were also checked);
- plant capacity for Phase 1 of 108 kt/d (including 75% ramp-up for first year of production);
- plant capacity for Phase 2 increases to 162 kt/d (including 75% ramp-up of expansion capacity in the first year of expansion, Year 10); and
- mine operations of 365 days per annum.

16.5.1 Pre-Production

Two years of pre-production mining are required to carry out the following tasks:

Pioneering: this includes the development of 13 km of mining roads which are mostly constructed of fill from overburden or waste. The roads will connect the upper elevation of the Mine Phase 1 pit to the tailings facility starter dams, the overburden stockpile, the operational ore stockpile, and the primary crusher dump area. This work will also create a platform for Mine Phase 1 and 2 which will be sufficient to allow the operation of the mining equipment.

Pre-Stripping: this includes the stripping of 38.6 Mt of waste rock from Mine Phase 1, which will expose sufficient ore to allow continuous ore delivery to the concentrator upon start-up. During this period, 14.8 Mt of ore will be transported to the ore stockpile for temporary storage.

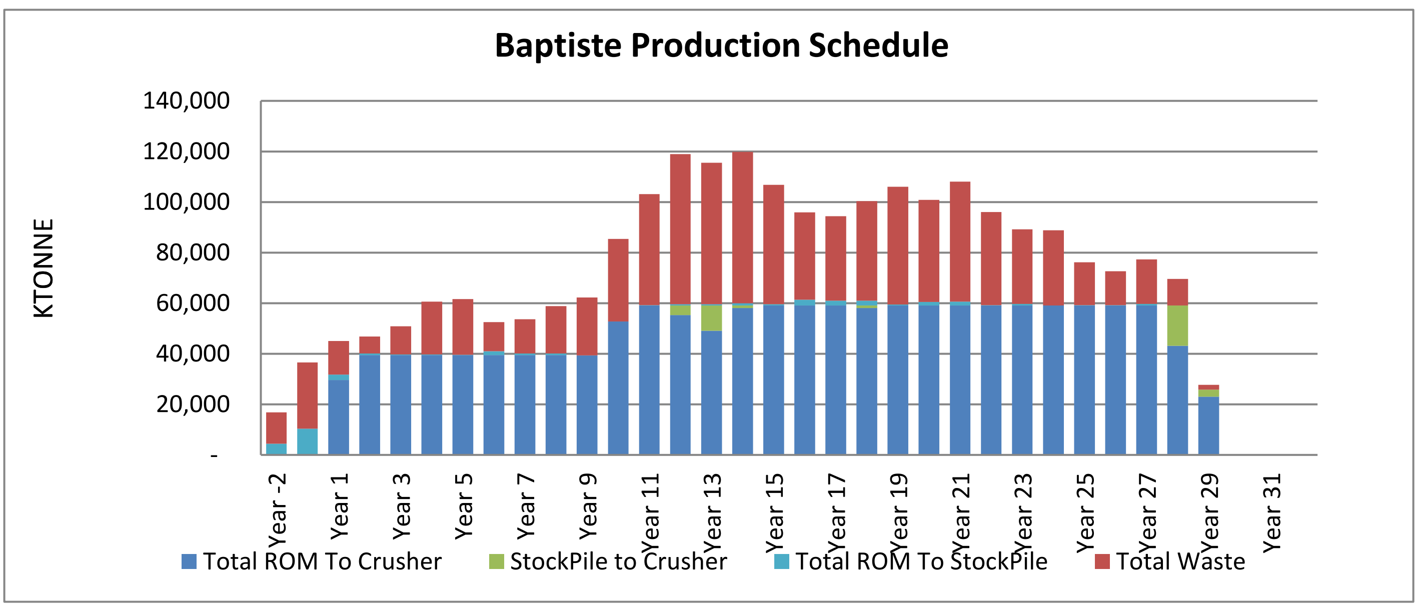
16.5.2 Production

Over the project's 29-year mine life, the concentrator will be fed with ore directly from the mine, and in the final year from the operational ore stockpile.

The production of the first year is 29.6 Mt (82 kt/d), representing 75% of the concentrator's nameplate capacity. Throughput is restricted this year as the plant commissioning is completed and throughput is ramped-up. In Years 2 through 9, the full 39.4 Mt (108 kt/d) of ore are delivered to the primary crusher. During this period, the mine ramps-up from approximately 15 Mt in Year -2 up to 50 in Year 3. Then, from Year 4 through 9 the mine moves an average of 60 Mt per annum.

In Year 10, a third grinding line will be commissioned, and the average annual concentrator throughput will increase to 59.1 Mt per annum (162 kt/d). During Years 10 through 29, the mine reaches a peak mining capacity of 120 Mt per annum, with an average mine capacity of 96 Mt per annum. Total material moved by the mine per annum is presented in Figure 16-14.

Figure 16-14: Baptiste Mine Production Schedule



Source: TechSer, 2023.

Material delivered to each final destination is presented in Table 16-11.

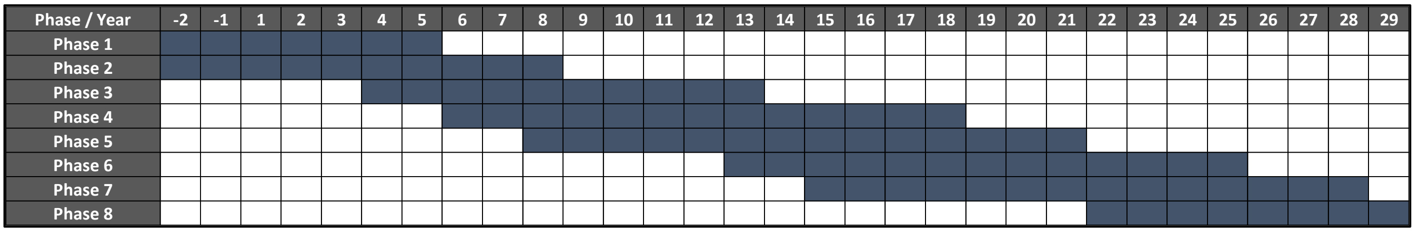
Table 16-11: Summary Table of Mine Production per Annum of Operation

Years of Operation			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29
Mining																																	
Total Material Moved	kt	**2,398,583**	16,879	36,600	44,990	46,884	50,921	60,584	61,664	52,474	53,678	58,787	62,303	85,455	103,070	118,951	115,560	119,873	106,748	95,873	94,410	100,281	106,046	100,893	108,046	96,048	89,165	88,771	76,235	72,677	77,368	69,614	27,735
Processing																																	
Total Ore Processed	kt	**1,487,721**	0	0	29,565	39,420	39,420	39,420	39,420	39,420	39,420	39,420	39,342	52,786	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,102	25,776
Ore Head Grade Nickel DTR	%	**0.130%**	0.00	0.00	0.139	0.137	0.145	0.137	0.137	0.146	0.138	0.118	0.12	0.127	0.135	0.140	0.125	0.130	0.137	0.140	0.129	0.124	0.121	0.124	0.120	0.126	0.134	0.133	0.131	0.137	0.121	0.111	0.116
Nickel Icp	%	**0.210%**	0.00	0.00	0.206	0.206	0.206	0.207	0.212	0.211	0.210	0.209	0.21	0.208	0.208	0.209	0.211	0.212	0.210	0.210	0.211	0.217	0.214	0.213	0.212	0.211	0.211	0.211	0.208	0.207	0.213	0.208	0.209
Cobalt DTR	%	**0.004%**	0.000	0.000	0.004	0.004	0.004	0.004	0.004	0.004	0.004	0.003	0.004	0.004	0.004	0.004	0.004	0.004	0.004	0.004	0.004	0.003	0.003	0.003	0.003	0.003	0.003	0.004	0.003	0.003	0.003	0.003	0.003
Fe DTR	%	**2.39%**	0.0	0.0	2.8	2.8	2.8	2.7	2.7	2.7	2.5	2.3	2.5	2.6	2.4	2.4	2.5	2.6	2.8	2.7	2.5	2.2	2.1	2.0	2.0	1.9	2.1	2.1	2.0	2.2	2.4	2.4	2.3
Recovery																																	
Nickel Recovery	%	**85.0%**	0.0	0.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0
Cobalt Recovery	%	**0.0%**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Concentrate Grade																																	
Nickel Conc. Grade	%	**63.0%**	0.0	0.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0	63.0
Ni Recovered	kt	**1,643**	-	-	35	46	48	46	46	49	46	40	40	57	68	70	63	65	69	70	65	62	61	62	61	63	67	67	66	69	61	56	25
Ni Conc Produced	dmt conc	**2,608,633**	-	-	55,406	72,864	76,960	72,864	72,918	77,811	73,396	62,919	63,537	90,733	107,940	111,371	99,324	103,872	108,898	111,610	102,515	98,846	96,771	98,925	96,053	100,760	107,143	105,946	104,590	109,376	96,452	88,591	40,239
Mining																																	
Total Waste Mined	kt	**876,191**	12,433	26,183	13,237	6,813	11,170	20,874	22,075	11,449	13,532	18,716	22,961	32,669	43,845	59,404	55,931	59,893	47,150	34,456	33,376	39,336	46,528	40,428	47,403	36,836	29,375	29,633	16,950	13,399	17,668	10,512	1,958
Overburden	kt	**95,847**	10,986	18,421	9,494	962	4,131	3,137	897	729	5,876	3,654	973	1,280	2,807	7,363	4,177	1,473	856	1,152	1,018	2,340	873	631	2,293	1,859	1,277	903	908	2,726	2,650	0	0
Class A	kt	**601,848**	1,155	2,261	3,742	5,602	3,700	11,413	15,511	8,704	7,194	10,677	13,976	22,711	20,545	30,676	38,682	51,757	45,764	33,304	32,358	33,161	32,195	25,739	33,818	25,862	21,262	20,248	12,319	10,021	15,018	10,512	1,958
Class B	kt	**178,496**	292	5,500	0	248	3,339	6,325	5,667	2,015	463	4,385	8,012	8,677	20,494	21,365	13,072	6,662	529	0	0	3,835	13,459	14,058	11,292	9,115	6,835	8,483	3,723	652	0	0	0
To Stockpile	kt	**34,671**	4,447	10,417	2,188	652	330	290	169	1,606	726	651	0	0	95	417	499	851	468	2,288	1,904	1,816	388	1,335	1,513	82	661	8	155	148	570	0	0
From Stockpile	kt	**34,671**	0	0	0	0	0	0	0	0	0	0	0	0	0	3,874	10,028	1,041	0	0	0	1,041	0	0	0	0	0	0	0	0	0	15,953	2,735
Mine to Mill	kt	**1,453,050**	0	0	29,565	39,420	39,420	39,420	39,420	39,420	39,420	39,420	39,342	52,786	59,130	55,256	49,102	58,089	59,130	59,130	59,130	58,089	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	43,148	23,042

Source: TechSer, 2023.

The Baptiste mining phase sequences are presented in Figure 16-15.

Figure 16-15: Mine Phase Schedule



Source: TechSer, 2023.

16.5.3 End-of-Period Maps

The end-of-period maps were prepared for the project to illustrate the mine sequencing and were based on the PFS mine plan.

16.5.3.1 Closure and Reclamation Planning

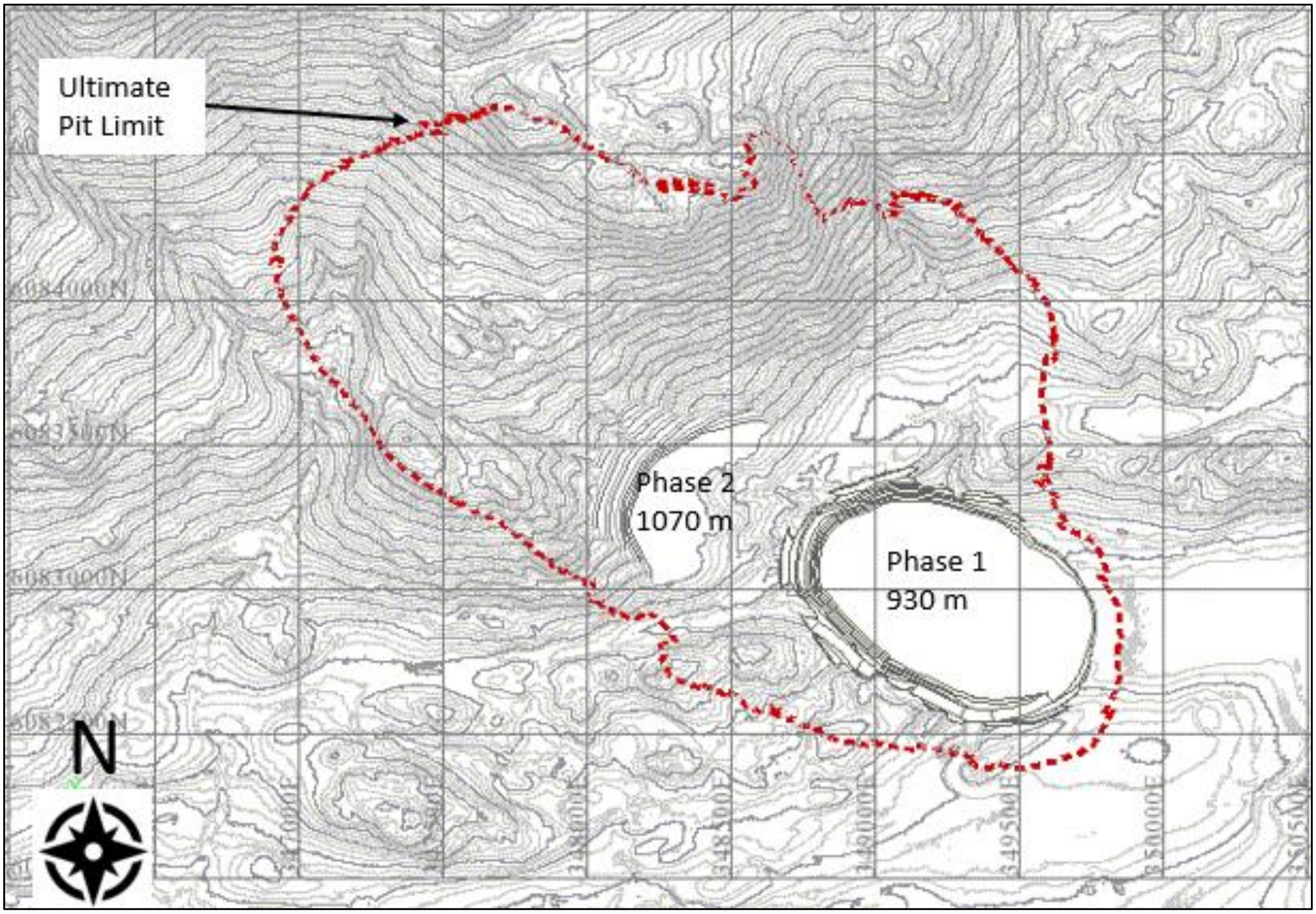
Year -1 of the Project corresponds to the end of the pre-stripping of Mine Phase 1 and 2.

Mine Phase 1 will be mined from elevation 980 masl (platform of the pioneering work) to elevation 930 masl and include a total of 5 full benches. The material mined in Mine Phase 1 totals 40.0 Mt, of which 14.8 Mt is ore sent to the operational ore stockpile and the remaining 25.2 Mt is waste sent to the tailings facility for starter dam construction. There is no ore sent to the primary crusher during this period as the concentrator is not yet in operation.

Phase 2 will be mined from elevation 1,120 masl (platform created during pioneering work) to elevation 1,070 masl and include a total of 5 full benches. The mined material from Mine Phase 2 totals 13.5 Mt, all of which is waste and to be used for tailings facility for dam construction.

No material is sent to the overburden stockpile during this period. The operational ore stockpile is at elevation 1100 masl (see Figure 16-16).

Figure 16-16: Plan at End of Year -1



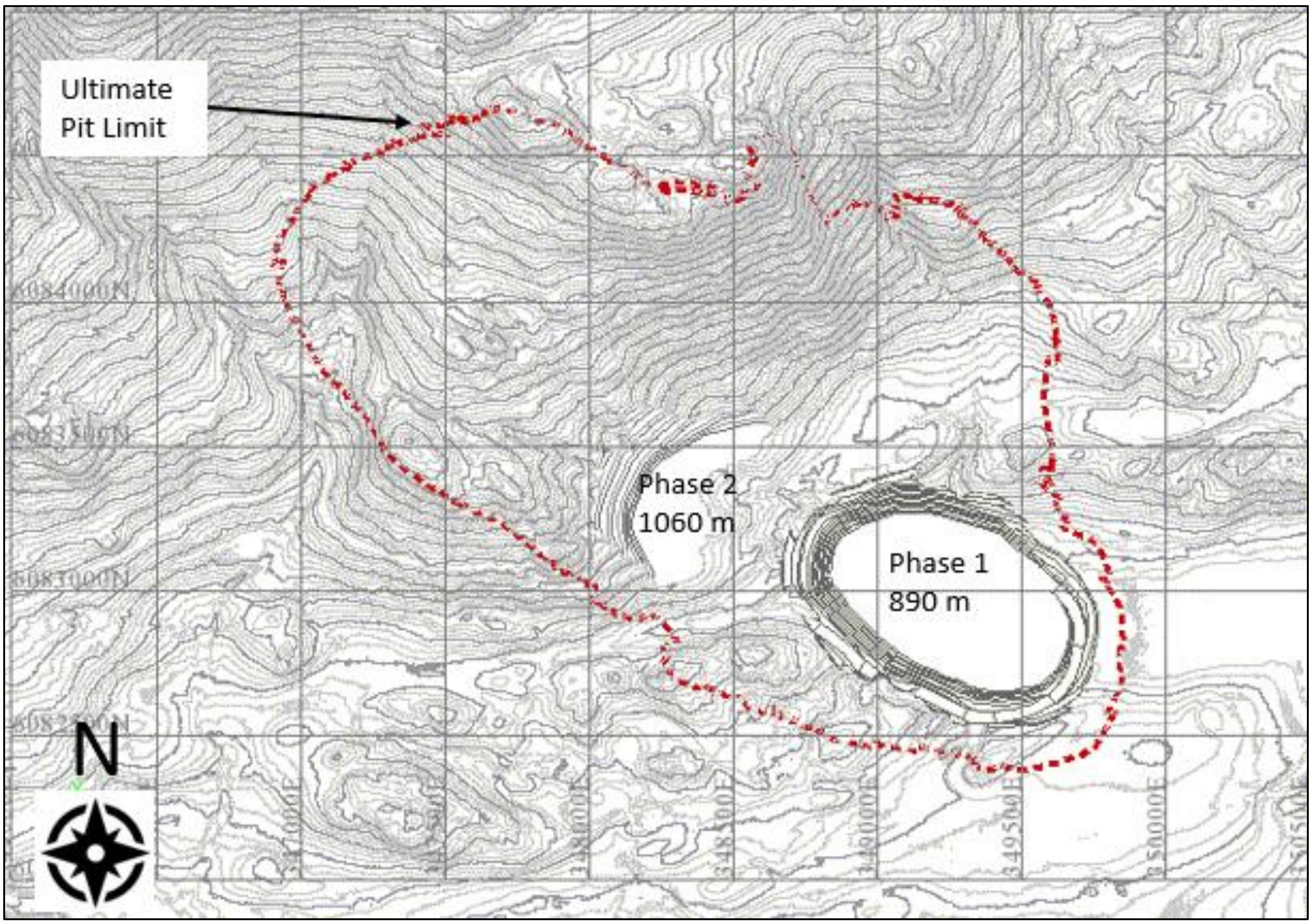
Source: TechSer, 2023.

16.5.3.2 Year 1

Year 1 of the project corresponds to the first year of ore production in which 29.6 Mt of ore is sent to the crusher, representing an average of 75% of the concentrator capacity (restricted during ramp-up).

Mine Phase 1 will be mined from elevation 930 masl to the elevation 890 masl with a total of 4 full benches mined. The mined material from Mine Phase 1 totals 44.8 Mt, of which 29.5 Mt of ore is sent directly to the crusher, 2.1 Mt is ore sent to the operational ore stockpile, and 13.2 Mt is waste sent to the tailings facility for dam construction. A small quantity of ore will be mined early from Mine Phase 2 and sent directly to the crusher (see Figure 16-17).

Figure 16-17: Plan at End of Year 1



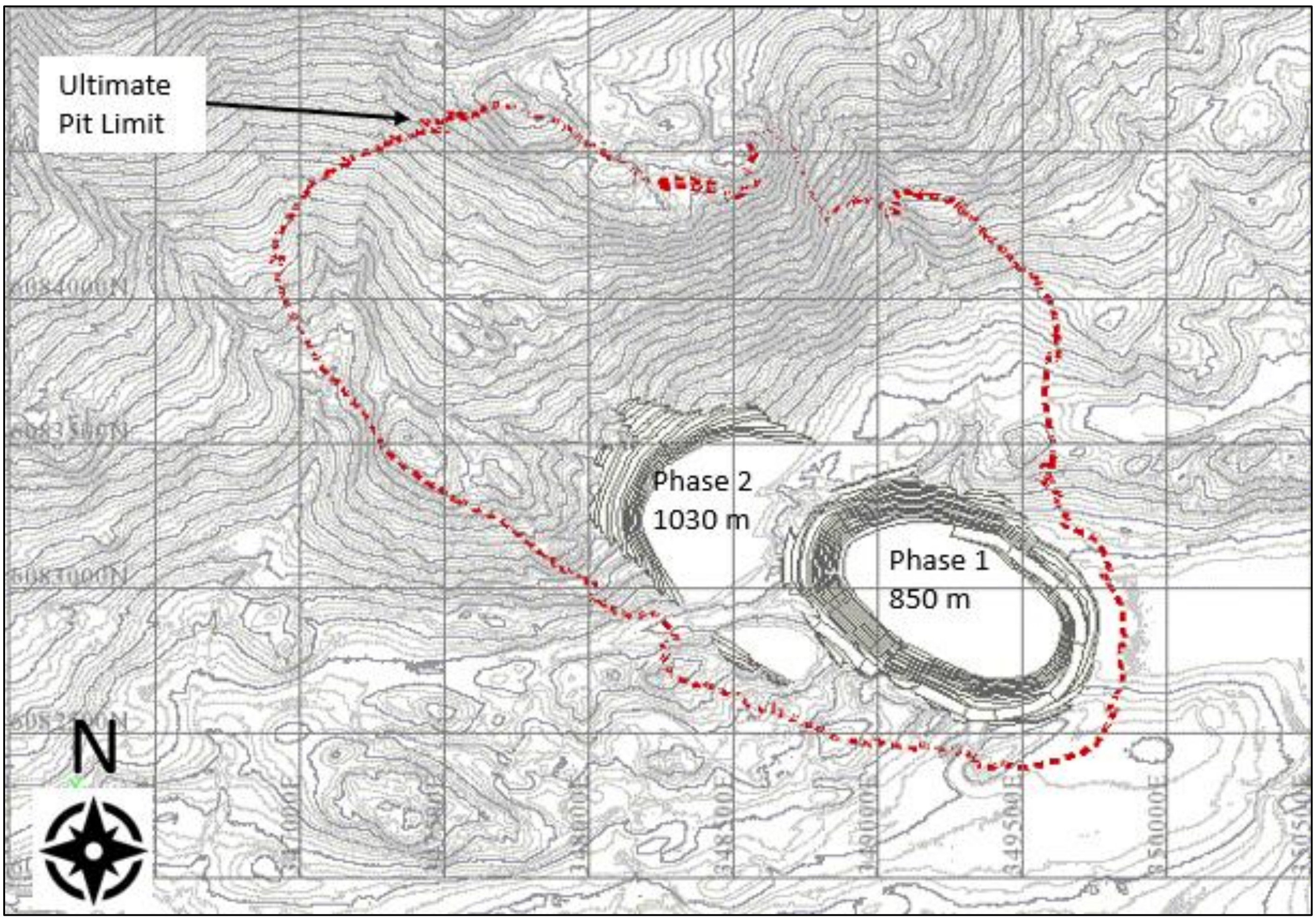
Source: TechSer, 2023.

16.5.3.3 Year 2

Year 2 of the project corresponds to the first operational year at full plant capacity, where 39.4 Mt of ore is sent to the crusher from Mine Phases 1 and 2.

Mine Phase 1 will be mined from elevation 890 masl to 850 masl with a total of 4 full benches mined. The material mined from Mine Phase 1 totals 30.6 Mt, of which 28.4 Mt is ore sent directly to the crusher and 2.2 Mt is used for tailings facility dam construction. Mine Phase 2 will be mined from elevation 1,060 masl to the elevation 1,030 masl. The material mined from Mine Phase 2 totals 16.4 Mt, of which 11.1 Mt is ore sent directly to the crusher and the rest is waste sent to the tailings facility for dam construction (see Figure 16-18).

Figure 16-18: Plan at End of Year 2

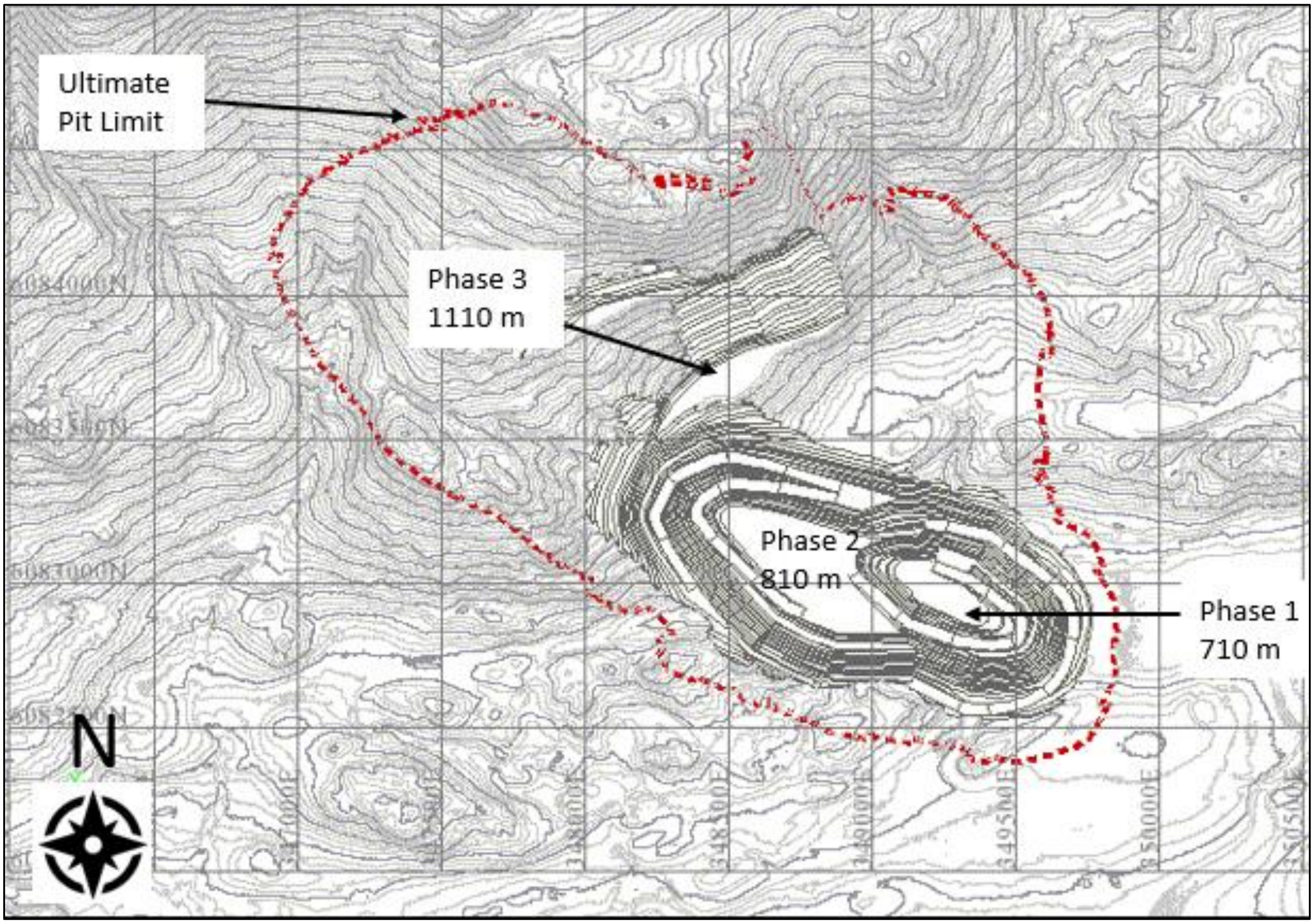


Source: TechSer, 2023.

16.5.3.4 Year 5

During this year, Mine Phase 1 will have been depleted. Mine Phase 2 will be mined from elevation 890 masl to elevation 810 masl with a total of 7 full benches mined. The material mined from Mine Phase 2 totals 49.5 Mt, of which 32.5 Mt is ore sent directly to the crusher and 17.0 Mt is waste sent to the tailings facility for dam construction. Mining of Mine Phase 3 will produce approximately 5.4 Mt of waste and supplement the rest of the ore to reach full concentrator production (see Figure 16-19).

Figure 16-19: Plan at End of Year 5



Source: TechSer, 2023.

16.5.3.5 Year 10

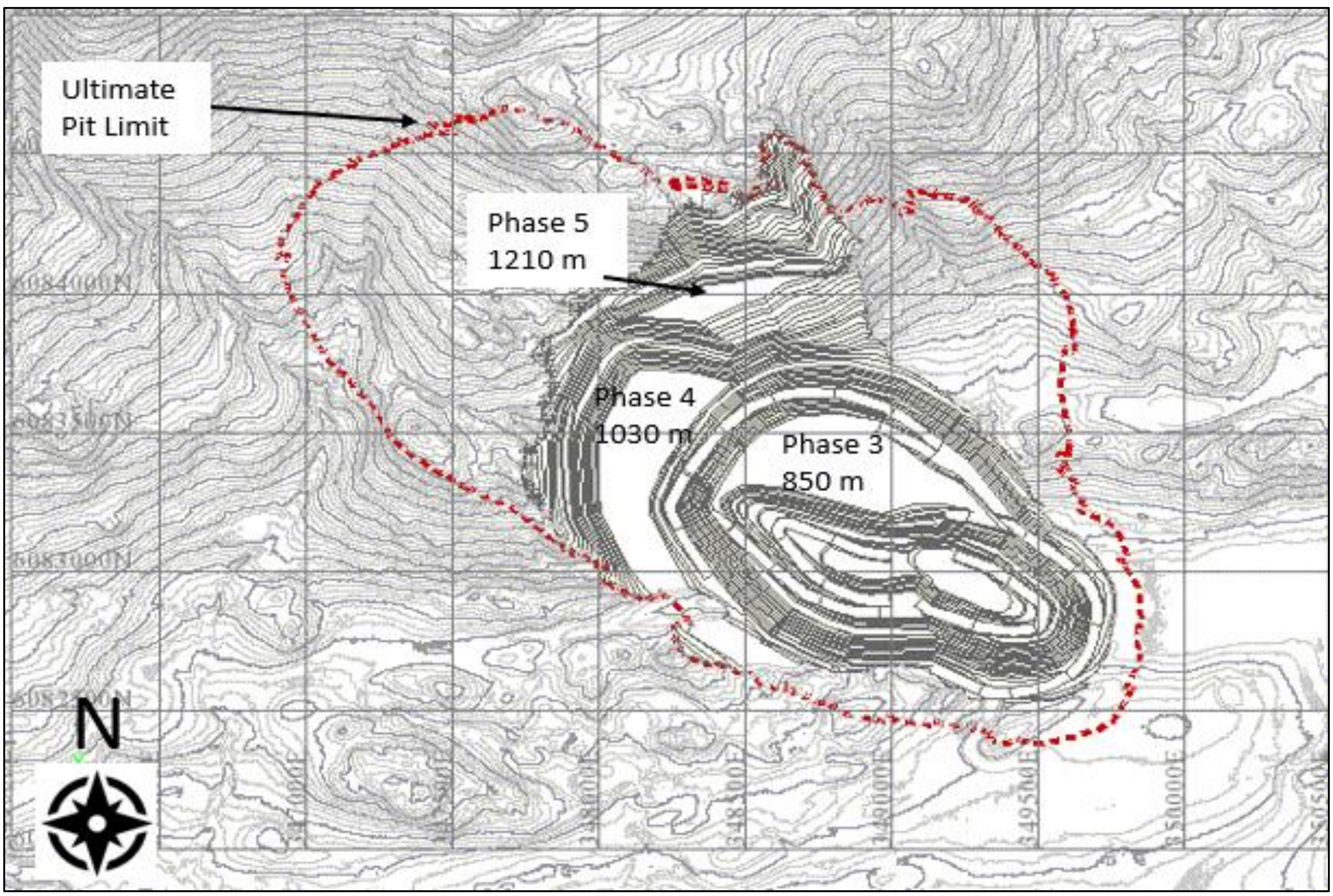
Year 10 of the project coincides with commissioning of the concentrator expansion, increasing total throughput capacity to 162 kt/d, where at this time Mine Phases 1 and 2 will have been depleted.

Mine Phase 3 will be mined up to the elevation 850 masl. The material mined from Mine Phase 3 totals 47.3 Mt, of which 38.1 Mt is ore sent directly to the crusher and 9.2 Mt is waste sent to the tailings facility.

Mine Phase 4 will be mined to elevation 1,030 masl. The material mined from Mine Phase 4 totals 26.4 Mt, of which 14.4 Mt is ore sent directly to the crusher and 12.2 Mt is waste sent to the tailings facility.

Mine Phase 5 will be mined to elevation 1,210 masl. The material mined from Mine Phase 5 totals 11.8 Mt, with almost all being waste which is sent to the tailings facility, and minor amounts of ore sent to the operational ore stockpile (see Figure 16-20).

Figure 16-20: Year 10



Source: TechSer, 2023.

16.5.3.6 Year 15

Mine Phases 4, 5, and 6 are in operation during Year 15.

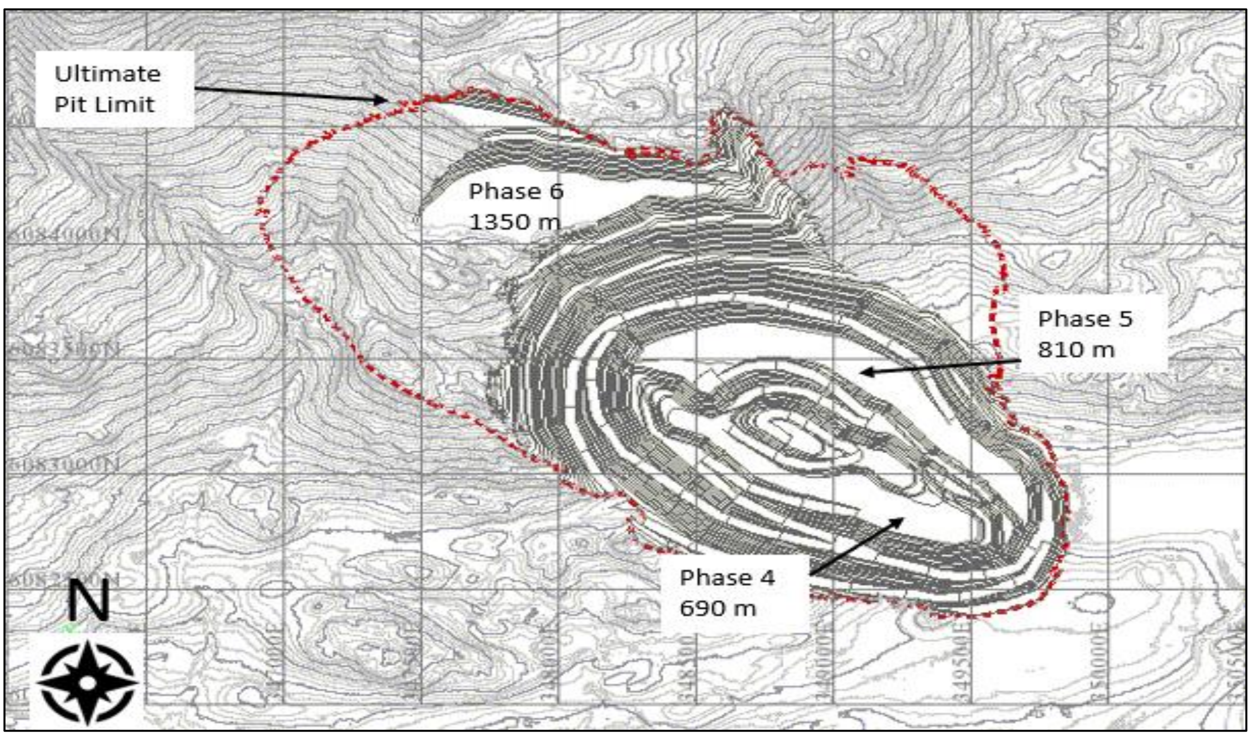
Mine Phase 4 will be operated to elevation 690 masl, with a total of 9 full benches mined. The material mined from Mine Phase 4 totals 51.9 Mt, of which 41.0 Mt is ore sent directly to the crusher and 10.5 Mt is waste sent to either the overburden stockpile or the tailings facility for dam construction.

Mine Phase 5 will be mined out to elevation 810 masl, with a total of 7 full benches mined. The material mined from Mine Phase 5 totals 23.7 Mt, of which 17.7 Mt is ore sent directly to the crusher and 5.9 Mt is waste sent to either the overburden stockpile or the tailings facility for dam construction.

Mine Phase 6 will be mined to elevation 1,350 masl, with a total of 7 full benches mined. The material mined from Mine Phase 6 totals 28.0 Mt, of which 27.6 Mt is waste sent to the overburden stockpile and the tailings facility for dam construction. Minor amounts of ore would be sent to the operational ore stockpile.

During this year, pioneering work commences in Mine Phase 7, with a small amount of waste mined from this phase (see Figure 16-21).

Figure 16-21: Plan at End of Year 15



Source: TechSer, 2023.

16.5.3.7 Year 21

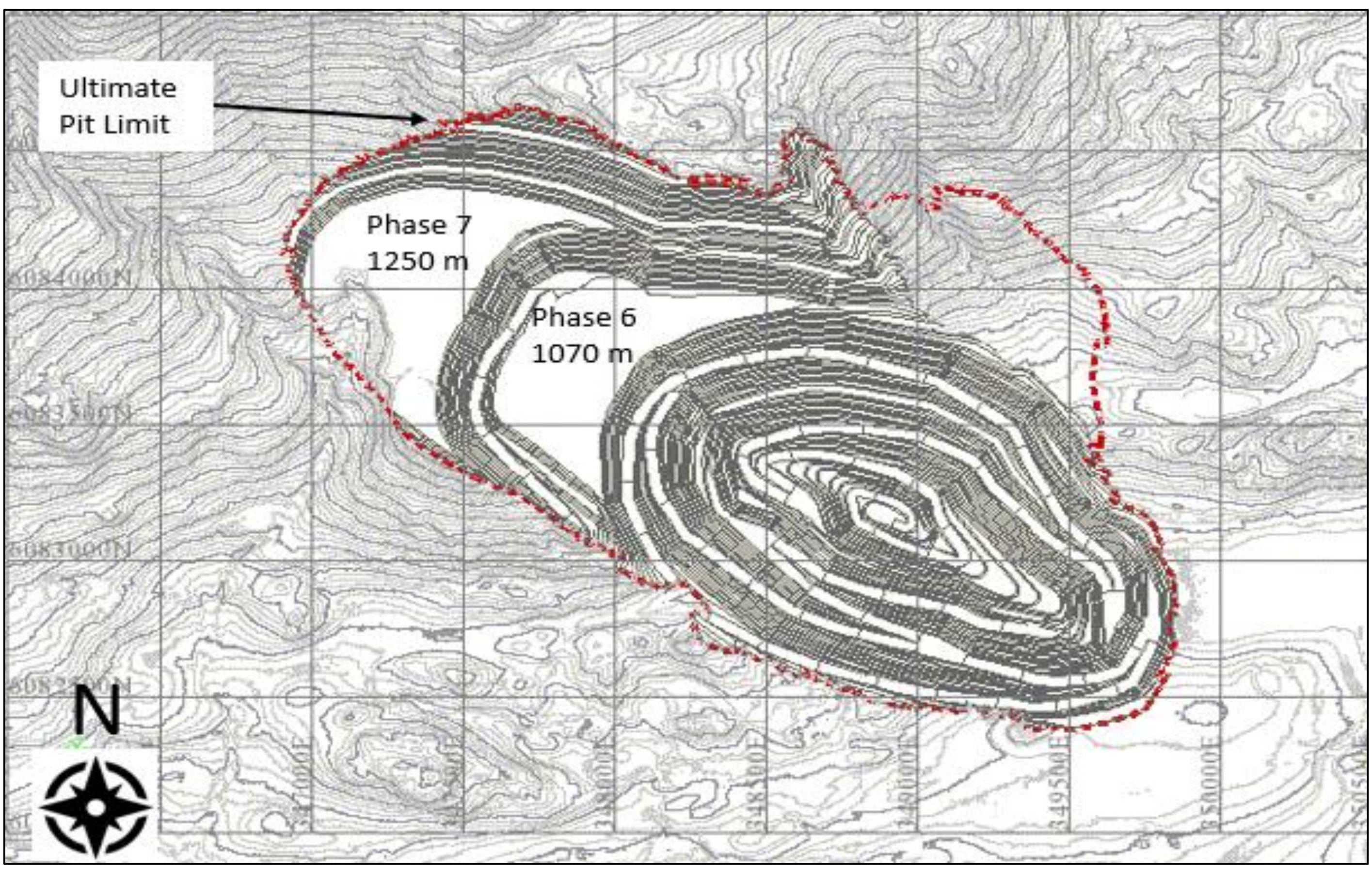
Mine Phase 5 will be depleted during this period. The last 3 benches of Mine Phase 5 will be mined with 1.7 Mt is ore being sent directly to the crusher and only 40 kt is waste being sent to the tailings facility for dam construction. This marks an important milestone as final bases of Mine Phases 4 and 5 are established, making this area available for in-pit backfill of waste or tailings. Refer to Section 18 for further discussion of waste rock management.

Mine Phase 6 will be mined to elevation 1,070 masl, with a total of 5 full benches mined. The material mined from Mine Phase 6 totals 64.9 Mt, of which 39.7 Mt is ore sent directly to the crusher, and 24.1 Mt is waste sent to either the overburden stockpile or the tailings facility for dam construction.

Mine Phase 7 will be mined to elevation 1,250 masl with a total of 4 full benches mined. The material mined from Mine Phase 7 total 36.4 Mt, of which 17.7 Mt is ore sent directly to the crusher, and 18.2 Mt waste sent to either the overburden waste dump or the tailings facility for dam construction.

During this period pioneering work commences in Mine Phase 8 (see Figure 16-22).

Figure 16-22: Plan at End of Year 21



Source: TechSer, 2023.

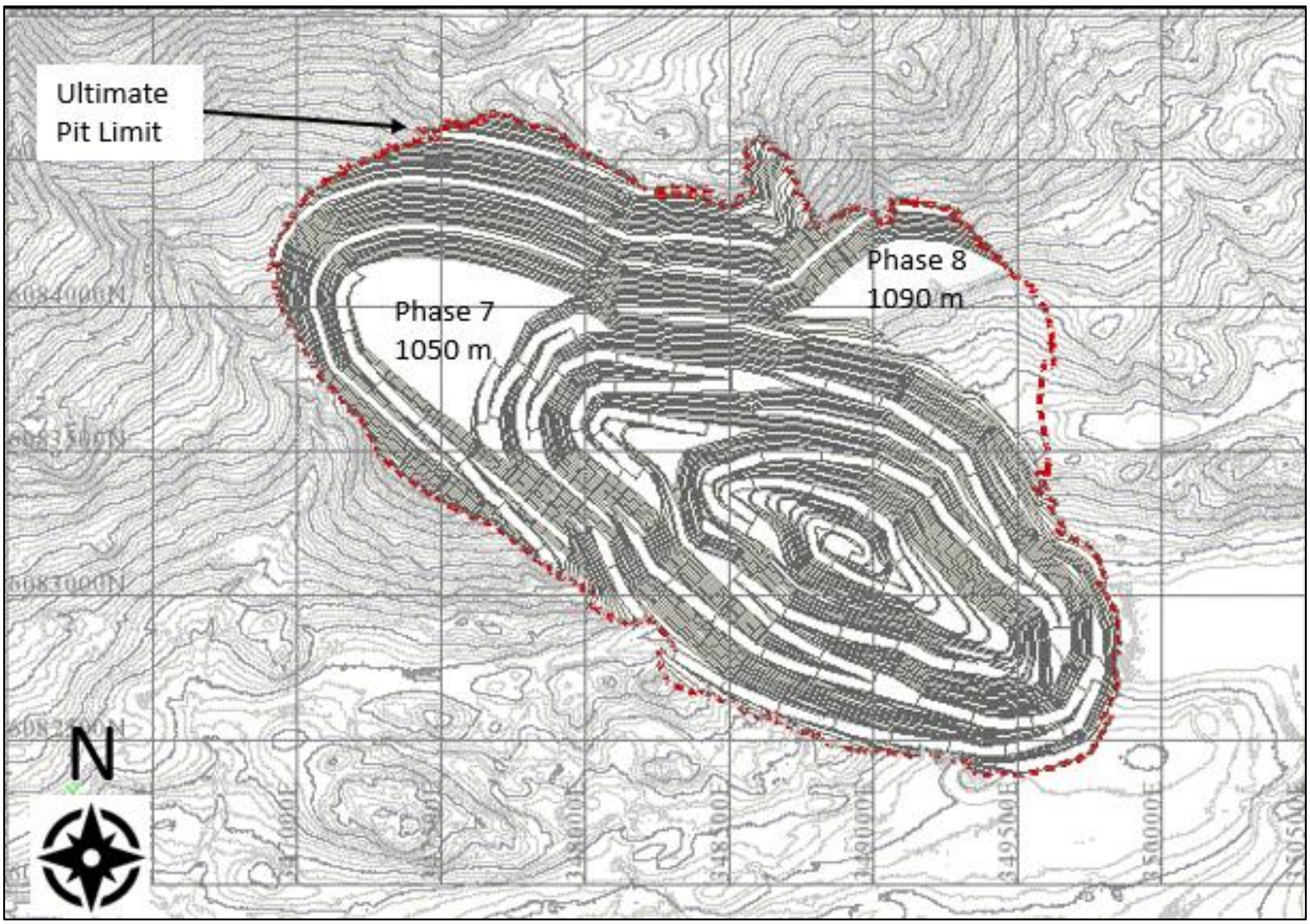
16.5.3.8 Year 25

Mine Phase 6 will be depleted during this period. The last 2 benches of Mine Phase 6 will be mined, with only 1.9 Mt of ore sent directly to the crusher and approximately 300 kt of waste sent to either the overburden stockpile or the tailings facility for dam construction.

Mine Phase 7 will be mined to elevation 1,050 masl, with a total of 6 full benches mined. The material mined from Mine Phase 7 totals 64.4 Mt, of which 54.1 Mt is ore sent directly to the crusher and 10.2 Mt is waste sent to either the overburden stockpile or the tailings facility for dam construction.

Mine Phase 8 will be mined to elevation 1,090 masl, with a total of 2 full benches mined. The material mined from Mine Phase 8 totals 9.7 Mt, of which 3.1 Mt is ore sent directly to the crusher area and 6.5 Mt is waste sent to either the overburden stockpile area or tailings facility for dam construction (see Figure 16-23).

Figure 16-23: Plan at End of Year 25



Source: TechSer, 2023.

16.6 Mining Operations

The Baptiste Deposit is a large, near surface, bulk mineable deposit that is well suited for conventional large-scale truck and shovel operation. The pit operations will work two 12-hour shifts per day with four crews on a two-week in, two-week out rotation. Engineering, geology and some operations supervisory or support positions will be on day-only 12-hour shifts that will also rotate every two weeks.

The following sections discuss the selection of equipment, with peak requirements for primary production equipment presented in Table 16-12.

Table 16-12: Peak Primary Production Equipment Counts

Equipment Type	Unit	Equipment Class
Blasthole Drill	mm	251
Blasthole Drill (pre-split)	mm	140
Truck	t	300
Hydraulic Shovel	m^3	42
Front-End Loader	m^3	19
Track Dozer	m^3	22
Grader	m	24
Rubber Tired Dozer	m^3	25

16.6.1 Drilling

Production drilling will be accomplished using a fleet of electric rotary blasthole drills. These units are capable of drilling 251 mm diameter holes on the designed 10 m benches via a single pass and have been selected because of their low unit operating cost and high productivity. A diesel drill will be leased during the 2 years of pre-production, before the pit is electrified.

The blast pattern size for ore was estimated by using a high-level fragmentation prediction model with the target of producing a fragmentation distribution curve with a P_{80} passing size of approximately 300 mm. Waste did not need to meet this size specification, so the blast pattern used for waste was expanded slightly. The drill pattern specifications are presented in Table 16-13.

Table 16-13: Drilling Parameters

Parameters	Unit	Ore	Waste
Hole Diameter	inches	9.875	9.875
	mm	251	251
Conversion factor	inches/mm	0.04	0.04
Bench Height	m/hole	10.0	10.0
Subgrade	m/hole	1.5	1.5
Hole Depth	m/hole	11.5	11.5
Burden	m	7.0	8.0
Spacing	m	7.0	8.0
Hole Volume	m^3/m	0.049	0.049
	m^3/hole	490	640
Hole Tonnage	t/hole	1,308	1,728
Drilling Yield	t/m	114	150
Pure Penetration	m/oph	40.0	40.0
Redrills/Ramps/Buffer	%	5.0%	5.0%
Effective Drill Productivity	t/oph	4,334	5,724

An opportunity exists to investigate autonomous drilling during the next stage of project planning.

16.6.2 Blasting

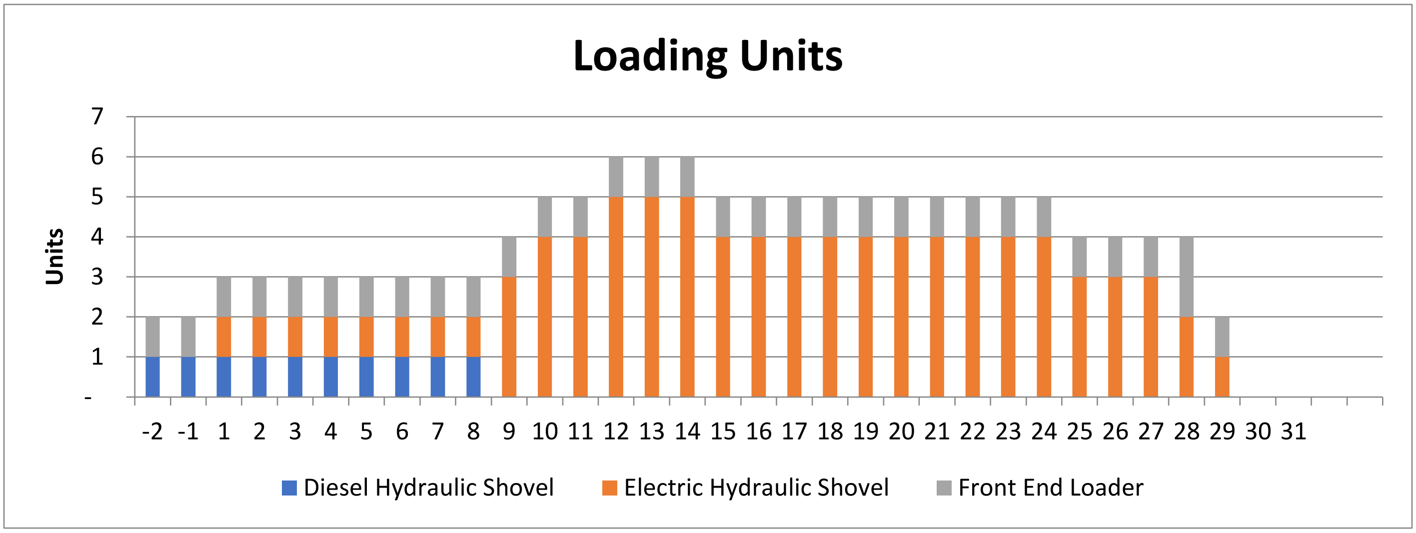
Blasting costs are estimated using budgetary quotations from local vendors. Unit costs were provided for bulk, packaged and initiating explosives. The powder factors used were 0.29 kg/t and 0.14 kg/t for ore and waste, respectively.

The vendors also quoted monthly service fees to cover the cost of capital and personnel in order to provide full blasting services. The quotes also included magazine rental, light vehicles, blasting trucks, site supervision, and the explosive plant lease.

16.6.3 Loading

Production loading will be performed by 42 m^3 electric hydraulic shovels, with 19 m^3 front-end loaders assisting with pit loading as well as ore rehandling from the operational ore stockpile. The equipment is well matched to the 10 m bench height. During the pre-production period, when the pit is not yet electrified, one diesel hydraulic shovel and one front-end loader will be operating. During the production period (Years 1 to 29) only electric shovels will be added to the fleet. The number of loading units required are presented in Figure 16-24.

Figure 16-24: Loading Fleet Opportunities



Source: TechSer, 2023.

16.6.4 Hauling

Truck performance depends heavily on travelling times, and therefore on haulage distance. The geometry of the deposit, the topography, the relative location of the crusher, and the various waste deposition locations result in a high percentage of downhill loaded hauling and flat hauling. The cycle time calculations were based on the hauling distances and speeds per road condition, as presented in Table 16-14 and Table 16-15.

Haul profiles were built for each year of operation using the end-of-period maps. A centreline was used for each destination, including the crusher and various ore and waste stockpiles. The set of centrelines were then imported into RPM's Talpac software where cycle times were calculated for each destination. Note that due to its size, the tailings facility was split into 4 areas to improve cycle time resolution, including the North Dam, West Dam, South Dam, and causeway.

Table 16-14: Truck Haulage Design Criteria

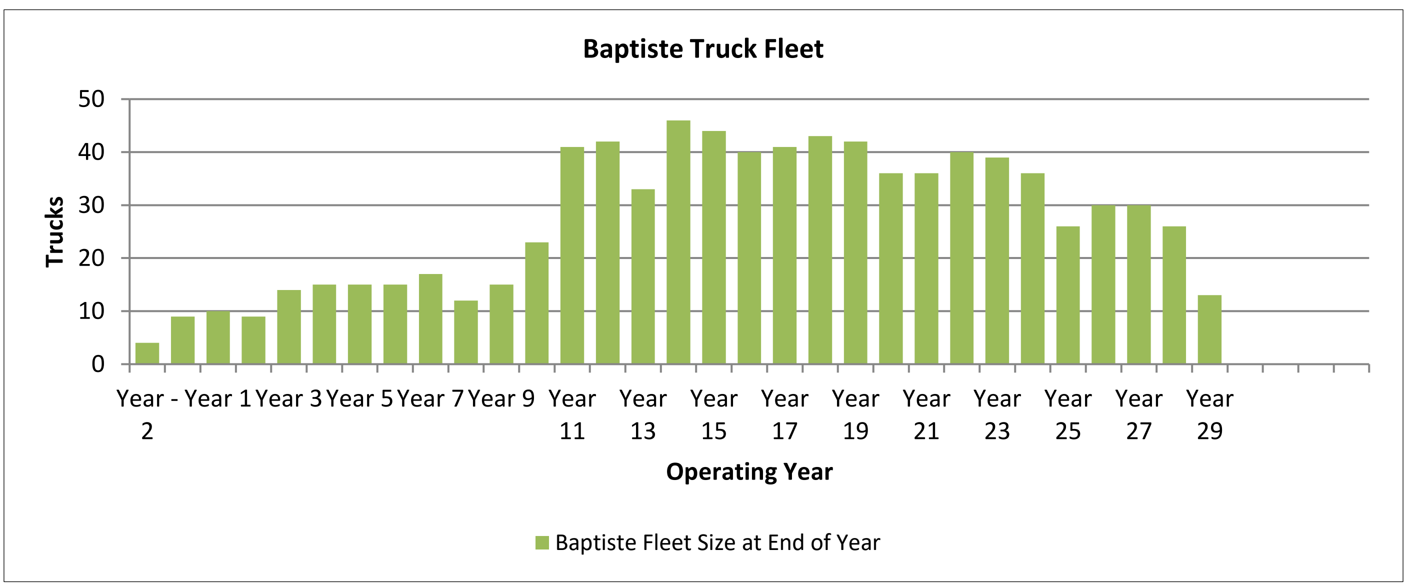
Baptiste	Unit	Speed
Surface Road and Ramp Speed limits		
Flat (<1%), Full	km/h	60
Flat (<1%), Empty	km/h	60
Uphill, Full	km/h	60
Uphill, Empty	km/h	60
Downhill, Full	km/h	60
Downhill, Empty	km/h	60
Steep Surface road and Ramp Speed Limits		

Baptiste	Unit	Speed
Steep road grade	%	10
Uphill, Full	km/h	15
Uphill, Empty	km/h	60
Downhill, Full	km/h	35
Downhill, Empty	km/h	35
Other Speed Limits		
Switchback, loaded truck	km/h	12
Switchback, empty truck	km/h	18
Bench and Lift road properties		
Bench maximum speed	km/h	35
Dump lift maximum speed	km/h	50

The number of hauling units required is presented in Figure 16-25 and was estimated using the PFS mine plan, cycle times from Talpac and the estimated asset utilization. The result was a Project Phase 1 peak requirement of 17 trucks in Year 7, and a Phase 2 peak requirement of 46 trucks in Year 14. Initially, 9 trucks are required for pre-stripping operations.

A truck service life of 100,000 h was used to develop a haul truck replacement schedule, with a major rebuild allowed at 72,000 hours. As such, the initial fleet of trucks will begin retiring in Year 15.

Figure 16-25: Haulage Fleet Quantities



Source: TechSer, 2023.

Table 16-15: Haulage Design Criteria

Hauling Indicators	Unit	Value
Mine Movement	Mt	2,399
Physical Availability	%	85
Utilization of Physical Availability	%	94
Operational Efficiency	%	82
Total Asset Utilization	%	65.5
Productivity	t/h	465
Average Equipment Count	units	27
Peak Equipment Count	units	46
Fuel Consumption per Truck	l/h	252
Operating Life for Tires	h	4,500
Average Travel Time, Ore	min	25.6
Average Travel Time, Waste	min	42.9

Truck performance considers loading by hydraulic excavators and front-end loaders, and separated by the two types of material, ore and waste. The parameters for the truck shovel matching for the project are presented in Table 16-16.

Table 16-16: Truck Shovel Match

Loading Equipment	Units	Hydraulic Shovel		Front-End Loader	
Loading Equipment Capacity	yd^3	42		25	
	m^3	32.1	32.1	19.1	19.1
Material Type		**ORE**	**WASTE**	**ORE**	**WASTE**
In-Situ Density	t/m^3	2.67	2.70	2.67	2.70
Swell Factor	%	*30*	*30*	*30*	*30*
Swelled Rock Density	t/m^3	2.05	2.08	2.05	2.08
Bucket Fill Factor	%	*90*	*90*	*96*	*96*
Bucket Capacity	t (wet)	59.4	60.0	37.7	38.1
Moisture	%	*2*	*2*	*2*	*2*
Bucket Capacity	t (dry)	58.2	58.8	36.9	37.3
Truck Model		**T274**			
Truck Capacity (initial)	t (wet)	305	305	305	305
Truck Capacity (initial)	t (dry)	299	299	299	299
Utilization of Truck Capacity	%	*100*	*100*	*100*	*100*
Truck Capacity (Projected)	t (dry)	299	299	299	299
Passes Required	quantity	5.1	5.1	8.09	8.00
Passes Assigned	**quantity**	**5**	**5**	**8**	**8**
Last Pass Efficiency	**%**	**94**	**98**	**99**	**90**

Loading Equipment	Units	Hydraulic Shovel		Front-End Loader	
Truck Capacity (Assigned)	t (dry)	299	299	299	299
Truck Capacity Used	%	100	100	100	100
Face Move Time	min/h	*10*	*10*	*10*	*10*
Shovel Set Up Moves	**min/h**	**10**	**10**	**10**	**10**
Total Cycle Time	**min**	**6.74**	**6.18**	**9.74**	**9.74**
Performance	**t/ophr**	**3,940**	**4,007**	**2,234**	**2,234**
Strip Ratio (W/O), including pre-stripping		0.59		0.59	
Maximum Operating Performance	**t/h**	**3,964**		**2,234**	
	t/d	**62,407**		**35,167**	
	t/a	**22,778,422**		**12,835,789**	

16.6.5 Auxiliary Equipment

The support equipment fleet will consist of track dozers, rubber tire dozers, graders, wheel loaders, excavators, and water trucks. Major tasks for support equipment include:

- cleaning and leveling of loading areas;
- maintaining and cleaning roads;
- blasting support/clean-up;
- waste rock facilities maintenance or roads and dumping areas;
- stockpile construction/maintenance;
- road building; and
- field equipment servicing.

A description of support equipment and key assumptions to calculate the fleet are presented in the following sections.

16.6.5.1 Track Dozer Fleet

The number of track dozers per annum is estimated from the following relationship: half a track dozer per loading unit and 1 track dozer for each storage facility and stockpile. For the waste placed in the tailings facility area, dozers are covered in the respective tailings facility cost center.

The main tasks for track dozers are to provide support in loading and dumping areas, construction of new roads, preparation for the drilling platform, and for supporting the overburden storage facility and operational ore stockpile areas. Track dozer will also support snow management in winter.

16.6.5.2 Rubber Tire Dozer Fleet

The number of rubber tire dozers per annum is estimated from the following relationship: 1 unit for every 3 pieces of loading equipment.

The main tasks for rubber tire dozers are to support removal of rocks along roads, safety berm construction, and generally support various areas as required.

16.6.5.3 Grader Fleet

The number of graders per annum required is estimated from the following relationship: 1 grader for every 8 hauling units per annum.

The main tasks for graders are to clean roads, assist with construction of new ramps in the mine, maintain roads from snow during winter, and other general support as required.

16.6.5.4 Excavator Fleet

The number of excavators per annum required is estimated from the following relationship: 1 excavator for every 200 kt/d of total mined material.

The main tasks for excavators are to support stability and clean wall slope, construction of trench drainage, and to provide general support in different areas as required.

16.6.5.5 Water Truck Fleet

The number of water trucks per annum required is estimated from the following relationship: 1 water truck for every 150 kt/d of total mined material.

The main tasks for water trucks are to support dust control along production routes for hauling operations, as well as dust control in the loading areas.

16.7 Mining Personnel

The mining personnel required for Baptiste is presented below and in Table 16-17.

An average of 128 personnel require to operate the fleet of drills, loading, hauling, and auxiliary equipment for project Phase 1, and average 272 personnel for project Phase 2, with an average of 221 personnel for LOM and a peak of 350 personnel in Year 14.

Personnel required to maintain the mining fleet average 77 for project Phase 1 and 174 for project Phase 2, with an average of 140 for LOM and a peak of 206 people in Year 14.

Personnel required for engineering, maintenance, and operations supervision averages 103 for project Phase 1 and 103 for project Phase 2 with an average of 103 for LOM.

Table 16-17: Average Minnig Personnel by Project Phase

Area	Phase 1	Phase 2
Total Mine Operations Personnel	128	272
Total Mine Maintenance Personnel	77	174
Total Mine Supervision and Technical Personnel	103	103
Total Mine Personnel	308	549

17 RECOVERY METHODS

17.1 Overview

The proposed process plant for the Baptiste Nickel Project will produce two nickel products. The main product, accounting for approximately 93% of nickel production, will be an awaruite (Ni_3Fe) concentrate produced by magnetic separation and froth flotation treatment. The second product, accounting for the remaining nickel production, will be a MHP coproduct produced from the precipitation of nickel leached into solution in the froth flotation and the subsequent flotation tailings leach circuits. The initial plant (Phase 1) is designed to process 108 kt/d of ore, which will be expanded for the start of Year 10 (Phase 2) to increase the total plant capacity to 162 kt/d of ore.

The selected process route is based on the metallurgical testwork performed since 2012 on samples from the Baptiste Deposit, as well as relevant industry benchmark data and Ausenco process experience. The Phase 2 design has taken the conservative approach of considering the expansion as an independent processing line. Future stages of study will evaluate the opportunity to integrate the Phase 2 processes into the original concentrator facility, in whole or in part. The major process steps included in the proposed flowsheet to produce the awaruite concentrate and MHP products in Phase 1 are listed below. Phase 2 will add a line of each of the process areas as independent process facility.

- One primary crusher and coarse ore stockpile.
- Two lines of SAB primary grinding.
- Two lines of rougher and rougher-scavenger low intensity magnetic separation (LIMS).
- First regrinding via horizontal ball mill in closed circuit with cyclones and gravity concentration.
- Cleaner and cleaner-scavenger LIMS.
- Second regrinding via stirred milling in open circuit.
- Recleaner and recleaner-scavenger LIMS.
- Awaruite flotation feed thickening.
- Awaruite rougher with conditioning and four-stage closed cleaner flotation.
- Awaruite concentrate dewatering via magnetic separation and press filtration.
- Awaruite concentrate briquetting, bagging, and outloading (into containers and on to trucks).
- Awaruite flotation tailings pre-leach thickening, acid leaching, and press filtration.
- Iron precipitation and removal.
- MHP product precipitation, thickening, and press filtration.
- MHP concentrate bagging and outloading (into containers and on to trucks).
- Magnesium precipitation and removal.

17.2 Plant Design

Key process design criteria for awaruite concentrate and MHP production are listed in Table 17-1.

Table 17-1: Summary of Major Process Design Criteria

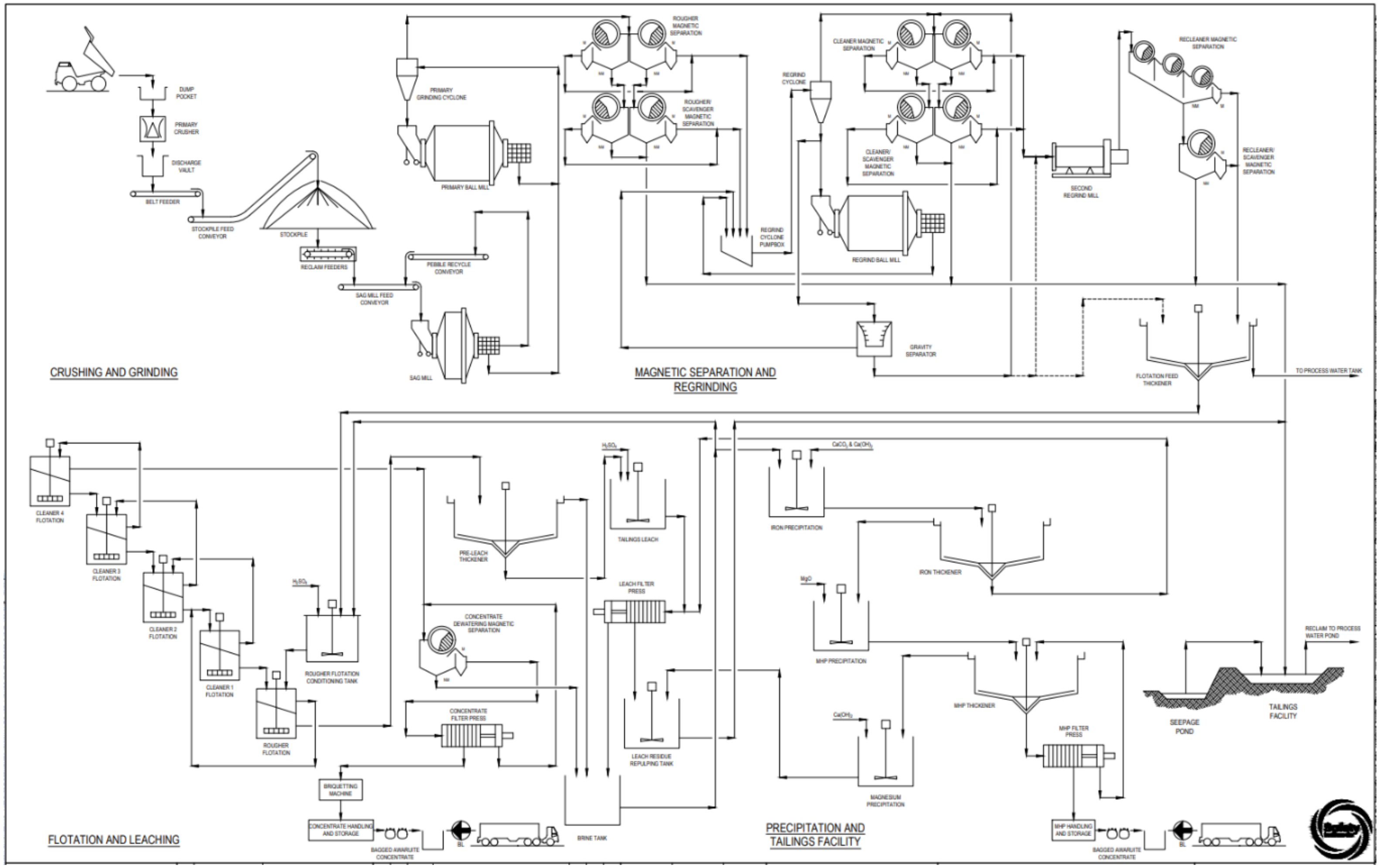
Criteria	Unit	Phase 1	Phase 2
Annual Throughput	Mt/a	39.4	59.1
Daily Throughput	kt/d	108	162
Operating Availability – Primary Crushing	%	75	75
Operating Availability – Grinding, Mag Sep, Flotation, Leach/Precipitation	%	92	92
Operating Availability – Filtration	%	85	85
Design Head Grade, DTR Ni	%	0.15	0.15
Design Head Grade, DTR Fe	%	2.84	2.84
Bond Crushing Work Index (CWi)	kWh/t	11.7	11.7
Bond Rod Mill Work Index (RWi)	kWh/t	17.6	17.6
Bond Ball Mill Work Index (BWi)	kWh/t	23.5	23.5
Axb	-	44	44
Bond Abrasion Index (Ai)	g	0.01	0.01
Crushing Circuit Feed Size, 80% Passing	mm	312	312
Crushing Circuit Product Size, 80% Passing	mm	85	85
Grinding Circuit Product Size, 80% Passing	µm	250	250
First Regrind Circuit Product Size, 80% Passing	µm	65	65
Second Regrind Circuit Product Size, 80% Passing	µm	18	18
Rougher/Scavenger Magnetic Separation			
DTR Ni Stage Recovery	% Circuit Feed	95.0	95.0
DTR Fe Stage Recovery	% Circuit Feed	95.0	95.0
Rougher Separation Stage Mass Pull	% Circuit Feed	15.2	15.2
Scavenger Magnetic Separation Stage Mass Pull	% Circuit Feed	4.5	4.5
Rougher Magnetic Tonnage per Magnet	t/h/m of roll	40	40
Rougher Pulp Rate per Magnet	m^3/h/m of roll	140	140
Scavenger Magnetic Tonnage per Magnet	t/h/m of roll	40	40
Scavenger Pulp Rate per Magnet	m^3/h/m of roll	140	140
Cleaner/Scavenger Magnetic Separation			
DTR Ni Stage Recovery	% Circuit Feed	99.3	99.3
DTR Fe Stage Recovery	% Circuit Feed	99.3	99.3
Cleaner and Scavenger Magnetic Separation Stage Mass Pull	% Circuit Feed	44.6	44.6
Cleaner Magnetic Tonnage per Magnet	t/h/m of roll	44	44
Cleaner Pulp Rate per Magnet	m^3/h/m of roll	110	110
Scavenger Magnetic Tonnage per Magnet	t/h/m of roll	44	44
Scavenger Pulp Rate per Magnet	m^3/h/m of roll	110	110

Criteria	Unit	Phase 1	Phase 2
Recleaner /Scavenger Magnetic Separation			
DTR Ni Stage Recovery	% Circuit Feed	99.7	99.7
DTR Fe Stage Recovery	% Circuit Feed	99.7	99.7
Recleaner and Scavenger Magnetic Separation Stage Mass Pull	% Circuit Feed	60.5	60.5
Recleaner Magnetic Tonnage per Magnet	t/h/m of roll	20	20
Recleaner Pulp Rate per Magnet	m^3/h/m of roll	84	84
Scavenger Magnetic Tonnage per Magnet	t/h/m of roll	20	20
Scavenger Pulp Rate per Magnet	m^3/h/m of roll	84	84
Awaruite Flotation			
Flotation Feed Thickener Unit Area	m^2/t/d	0.07	0.07
Rougher Conditioning Residence Time (Total)	min	15	15
Rougher Residence Time	min	36	36
Rougher Lip Loading	t/m/h	1	1
Rougher Froth Carrying Capacity	t/m^2/h	1	1
Cleaner Residence Time, Each	min	9	9
Cleaner Lip Loading, Each	t/m/h	1	1
Cleaner Froth Carrying Capacity, Each	t/m^2/h	1	1
DTR Ni Dissolution Loss	%	2.2	2.2
Cleaner Concentrate DTR Ni Recovery	% Flotation Feed	85.4	85.4
Cleaner Concentrate DTR Fe Recovery	% Flotation Feed	1.8	1.8
Cleaner Concentrate DTR Ni Grade	%	60.0	60.0
Awaruite Concentrate Dewatering			
Dewatering Magnetic Tonnage per Magnet	t/h/m of roll	32	32
Dewatering Pulp Rate per Magnet	m^3/h/m of roll	65	65
Concentrate Filter Press Filtration Rate	kg/m^2/h	210	210
Awaruite Tailings Leach, MHP Precipitation, and Purification			
Pre-Leach Thickener Unit Area	m^2/t/d	0.08	0.08
Leach Residence Time	hr	8	8
Leach Filtration Rate	kg/h/m^2	300	300
Iron Thickener Unit Area	m^2/t/d	0.4	0.4
MHP Thickener Unit Area	m^2/t/d	1	1
MHP Filter Press Filtration Rate	kg/h/m^2	150	150
Nickel Extraction	% Leach Feed	50.0	50.0
Dissolved Nickel Recovery to MHP	% Dissolved Ni	89.0	89.0

17.3 Process Flowsheet

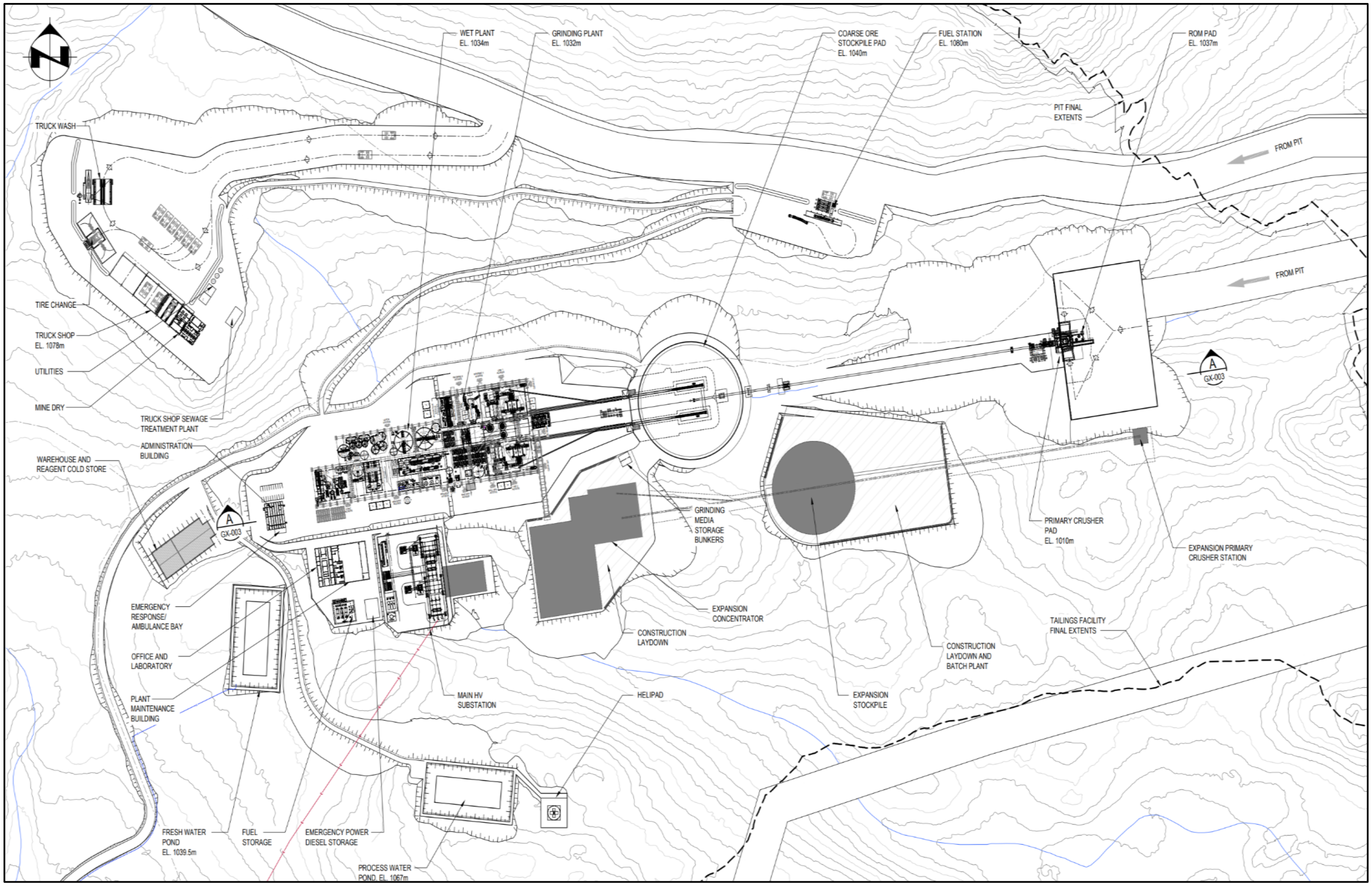
The Baptiste overall process flow diagram for Phase 1 is provided in Figure 17-1. The design includes a single gyratory crusher feeding a coarse ore stockpile. Ore will be fed from the stockpile to two parallel grinding circuits, each with a semi autogenous grinding (SAG) mill feeding a ball mill. Each grinding circuit will feed dedicated rougher and rougher-scavenger magnetic separation circuits. Rougher and rougher-scavenger magnetic concentrate will be combined and processed through a single line of circuits as shown on in Figure 17-1. The plant layout diagram is shown in Figure 17-2. Details of the Phase 1 flowsheet for 108 kt/d and major process equipment are described in the following sections. A complete, independent, processing line will be installed for Phase 2, increasing the total processing capacity to 162 kt/d. Only the initial phase of the process plant is described in this section as the expansion follows the same flowsheet.

Figure 17-1: Overall Process Flow Diagram



Source: Ausenco, 2023.

Figure 17-2: Processing Plant Overall Layout

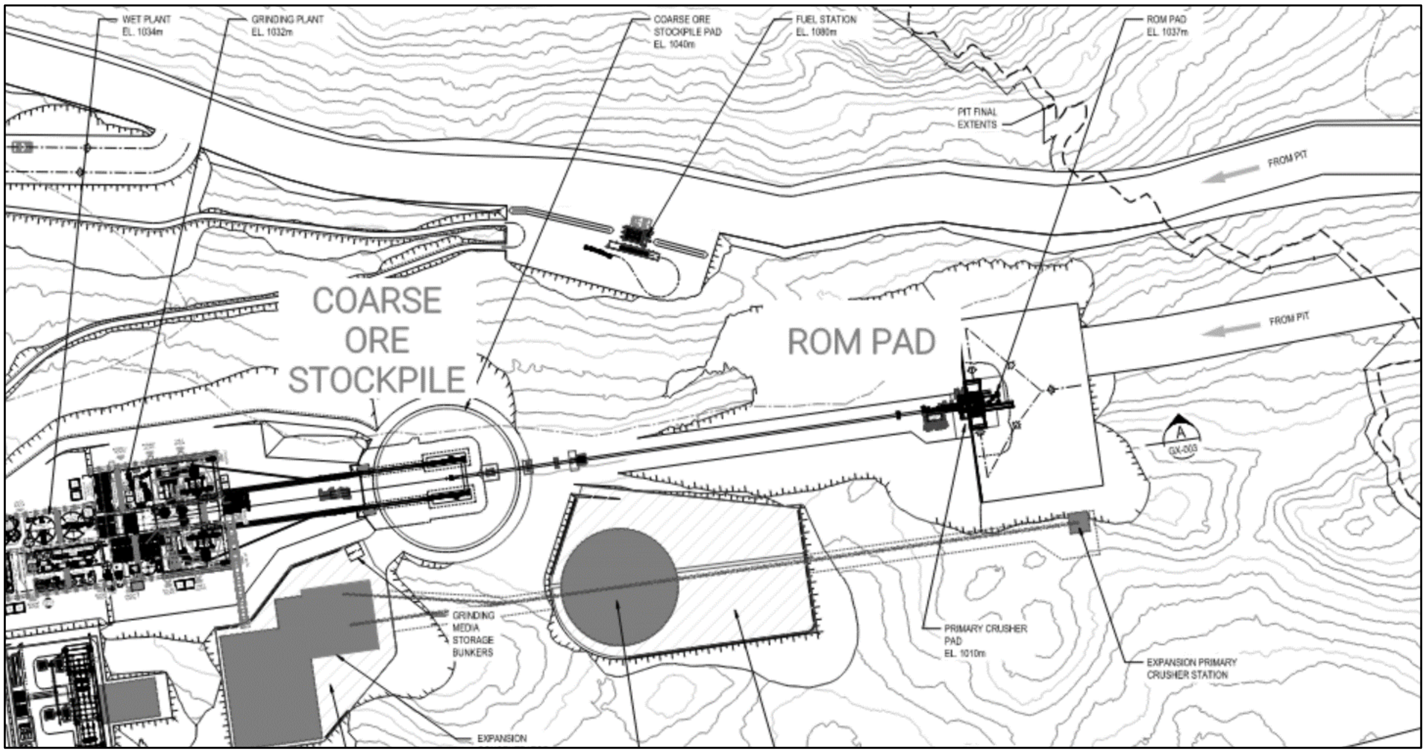


Source: Ausenco, 2023.

17.3.1 Primary Crushing and Coarse Ore Stockpile

The primary crushing circuit is designed assuming operating time of 6,570 hours per year, an equivalent of 75% availability. The Phase 1 crushing circuit is designed for a nominal throughput of 6,000 t/h. The primary crushing and coarse ore stockpile are depicted in Figure 17-3.

Figure 17-3: Primary Crushing and Coarse Ore Stockpile Layout



Source: Ausenco, 2023.

The Phase 1 primary crushing circuit will consist of a single gyratory crusher and a covered coarse ore stockpile complete with two reclaim tunnels. Run of mine (ROM) ore will be directly dumped from haul trucks into the primary crusher dump pocket. The dump pocket will have a capacity of 1.5 truckloads and will be equipped with a hydraulic rock breaker to break-up any oversized rocks. The ore will flow by gravity into the primary crusher, which will be choke-fed to the greatest extent possible.

The primary gyratory crusher is designed to reduce the ROM ore from an 80% passing feed size (F_{80}) of 312 mm to an 80% passing product size (P_{80}) of 85 mm. The crusher will be equipped with a dust collection system and collected material will be discharged onto the stockpile feed conveyor. The crushed ore will drop to a 450 t capacity vault below the crusher where it will be reclaimed by a variable speed belt feeder that will transfer the ore onto the stockpile feed conveyor. The covered stockpile is designed to provide surge capacity and will be equipped with a dust collection system.

Two reclaim lines with two apron feeders each will move the stockpiled ore to their respective SAG mill grinding feed conveyors.

The area includes the following major equipment and facilities:

- One ROM crusher dump pocket, 450 t live capacity.
- One primary gyratory crusher of 1.5 m gape x 2.8 m mantle diameter, 1,372 kW.
- One rock breaker.
- One covered coarse ore stockpile, 54 kt live capacity, 270 kt total stockpile capacity.
- One primary crusher maintenance crane.
- Associated belt feeder, belt conveyors and dust collection systems.

17.3.2 Primary Grinding

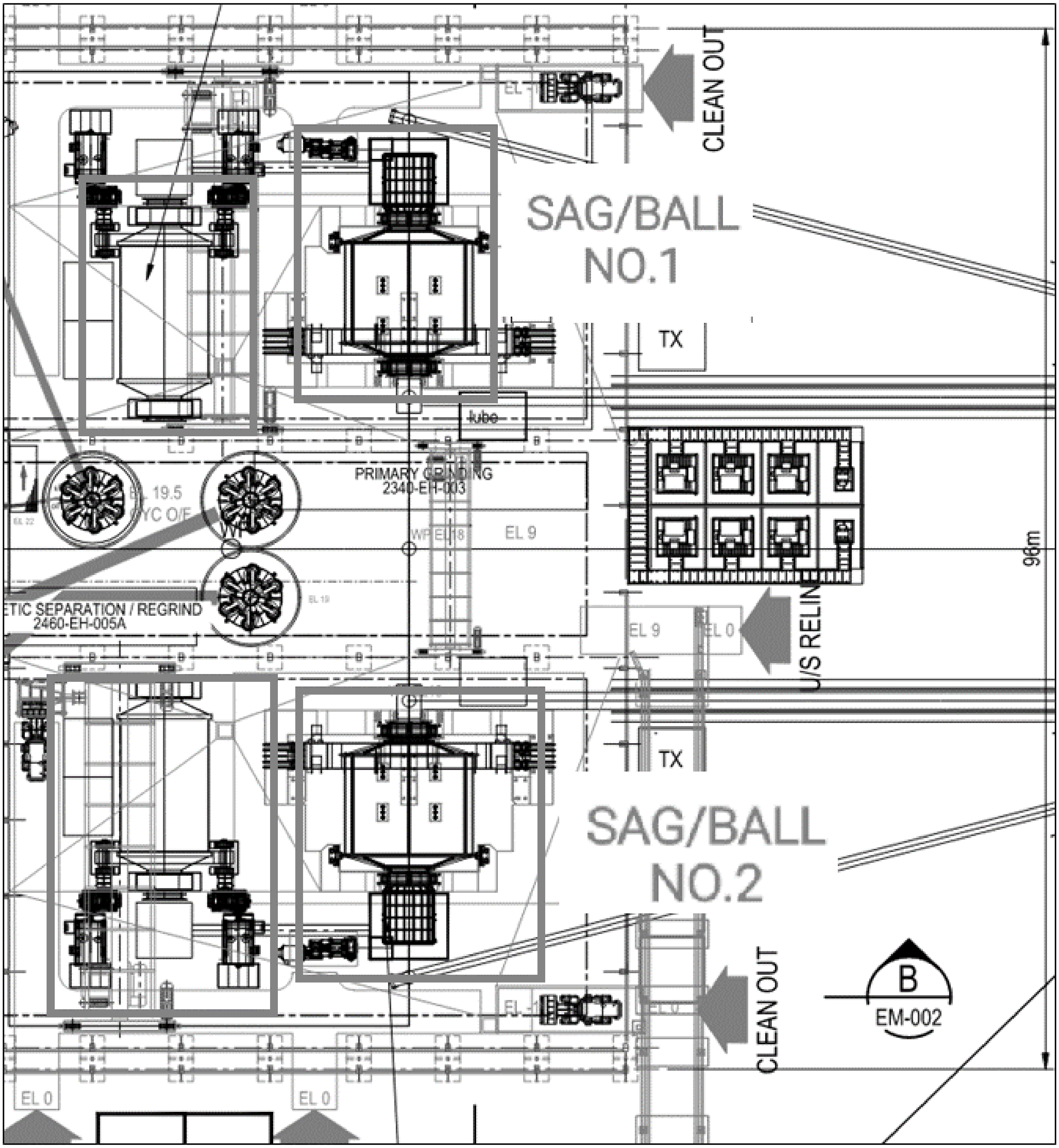
Two parallel SAB primary grinding lines are designed for an operating time of 8,060 hours a year, equivalent to 92% availability, with a combined nominal throughput of 4,891 t/h, or 2,446 t/h per line. Each grinding line will consist of a high-aspect gearless drive SAG mill followed by a dual pinion ball mill in a closed circuit with cyclones. The circuit is sized based on a grinding circuit feed size F_{80} of 85 mm and a target grind product size P_{80} of 250 µm. The grinding circuit layout is depicted in Figure 17-4.

Reclaimed ore from the coarse ore stockpile will report to the two parallel SAG mill feed conveyors, which transports the ore to the respective grinding lines. The SAG mills are designed to operate with a pulp density of 70% by weight and 125 mm grinding media. The SAG mill product will be discharged through a SAG mill trommel screen to separate the oversized ore (pebbles), while the undersize ore will gravity flow to the primary cyclone pump box and be pumped to the primary cyclone cluster. The pebbles pass under a tramp magnet to remove any foreign material that will be disposed of in a removable scats bin. The pebble recirculation conveyors are designed to transport up to 30% of the fresh feed rate back to each respective SAG mill feed conveyor.

The primary cyclone underflow will gravity flow to the dedicated ball mill. The ball mill feed will have a pulp density of 54% by weight. The ball mills will utilize 60 mm steel balls for grinding media that will be added using a common ball mill charging kibble for both ball mills. The discharge from the ball mill will be screened through the ball mill trommel with oversize reporting to a removable scats bin and undersize gravity flowing to the primary cyclone pump box and will be pumped back to the primary cyclone cluster for classification. The primary cyclone cluster is sized for a 250% nominal circulating load. The primary cyclone overflow, at a nominal pulp density of 35% by weight, will gravity flow to the magnetic separation circuit.

Each grinding line, SAG and ball mill, will have a dedicated overhead crane for maintenance.

Figure 17-4: Primary Grinding Area Layout



Source: Ausenco, 2023.

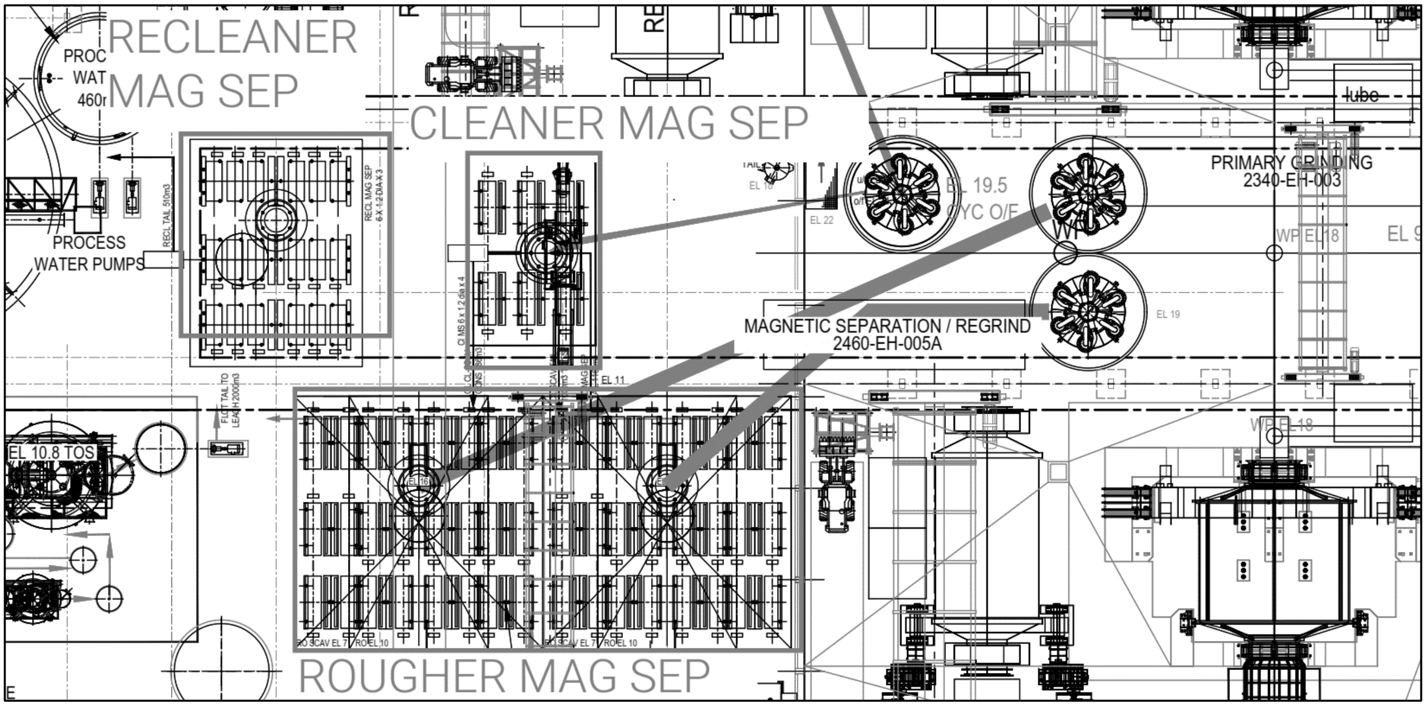
Each primary grinding line will include the following major equipment and facilities:

- One SAG mill, 11.6 m diameter by 9.1 m effective grinding length (EGL), fitted with a 25 MW gearless mill drive, variable speed, and 5 m diameter x 5.75 m length discharge trommel screen.
- One dual pinion ball mill, 7.9 m diameter by 13.1 m EGL, fitted with two 8.4 MW wound rotor induction (WRIM) mill drives, fixed speed, a liquid resistance starter (LRS) and discharge trommel screen.
- One primary ball mill cyclone cluster, including 18 operating and 2 standby cyclones, 650 mm diameter each.
- One crane per grinding line, 110 t capacity 36 m bridge span.
- One crane per primary cyclone line, 110 t capacity 20 m bridge span, shared with cleaner/cleaner-scavenger and recleaner/recleaner-scavenger LIMS.
- Associated feeders, conveyors, and pumps.

17.3.3 Magnetic Separation and Regrind

The magnetic separation and regrind circuits are designed for an operating time of 8,060 hours a year, equivalent to 92% availability, and will include three separate stages of magnetic separation via LIMS, integrated with two intermediate regrind stages utilizing horizontal ball milling and stirred milling technologies. The overall circuit will produce a magnetic concentrate in preparation for further flotation treatment. The magnetic separation circuit will recover approximately 94% of the DTR Ni and DTR Fe into approximately 5% of ore feed mass. The layout for the Phase 1 magnetic separation and regrind circuits are depicted in Figure 17-5.

Figure 17-5: Magnetic Separation and Regrind Area Layout



Source: Ausenco, 2023.

The Phase 1 magnetic separation and regrind circuits will consist of two lines of rougher and rougher-scavenger LIMS, followed by a single first regrind ball mill circuit, a single cleaner and scavenger LIMS circuit, a single second regrind stirred milling circuit, and a single recleaner and scavenger LIMS circuit.

All non-magnetic tailings from each LIMS circuits will be sent to the combined tailings pump box and transferred to the tailings facility. For the first seven years of operation, tailings will gravity flow from the tailings pump box to the tailings facility. In Year 8 a tailings pumping system will be installed as the tailings embankment elevation will have been raised to an elevation beyond which gravity flow is not possible. Tailings will be pumped to the tailings facility until Year 27. In Year 28, tailings will be pumped to the open pit, where it will be deposited through to mine closure.

17.3.3.1 Rougher and Rougher Scavenger Magnetic Separation

Overflow streams from the two primary cyclone clusters will gravity flow to their respective distributor. Each distributor will split and feed into 12 individual rougher LIMS, for a total of 24 rougher LIMS. Non-magnetic tailings from each of the rougher LIMS will directly gravity flow to a dedicated scavenger LIMS to recover any remaining magnetic minerals. The magnetic concentrate from all rougher and scavenger LIMS will be combined and gravity flow to the first regrind mill cyclone feed pump box. The non-magnetic tailings will gravity flow to the combined tailings pump box for discharge to the tailings facility.

Each magnetic separation line will include the following major equipment and facilities:

- Twelve (12) rougher LIMS each with a single roll of 1.2 m diameter and 4 m width and magnetic field strength of 1,200 gauss.
- Twelve (12) rougher scavenger LIMS each with a single roll of 1.2 m diameter and 4 m width and magnetic field strength of 1,800 gauss.
- One (1) magnetic separation area crane, 20 t capacity, 36 m bridge span.
- Associated feed distributors and pumps.

17.3.3.2 First Regrind

The first regrind circuit will include a single horizontal ball mill closed with classification cyclones to produce an overflow at a P_{80} of 65 µm to feed the cleaner magnetic separation circuit. The underflow from one cyclone will be treated in a continuous-type centrifugal gravity concentrator to recover liberated awaruite. At the plant design feed grades, approximately 50% of DTR Ni and 2.3% of DTR Fe in the gravity circuit feed is assumed to be recovered into the gravity concentrate. This gravity concentrate will then be pumped to the cleaner magnetic separation circuit for further upgrading; alternatively, it can be bypassed to the second regrind cyclone feed pump box or the flotation feed thickener as needed. The gravity tailings will return to the first regrind feed cyclone pump box.

The underflow from all other first regrind cyclones will be sent to the ball mill for regrinding using 25 mm steel balls. Steel balls will be received from the regrind mill media storage bunker and loaded into the ball mill using the common kibble also used for the primary ball mills. The first regrind ball mill discharge, at a pulp density of 75% by weight, will gravity flow through a trommel screen with the oversize portion collected in a removable scats bin. The trommel screen undersize will gravity flow to the first regrind cyclone pump box and will be pumped to the first regrind cyclones for reclassification.

A single overhead crane will be used for both the first and second regrind area.

The area will include the following major equipment and facilities:

- One dual pinion regrind ball mill, 7.9 m diameter by 13.1 m EGL, fitted with 2 (two) 8.4 MW WRIM mill drive motors, a LRS and discharge trommel screen (all drive components are common with the primary ball mills).
- One regrind ball mill cyclone cluster, including 15 operating and 2 standby cyclones, 500 mm diameter each.
- One, 190 t/h solids, centrifugal-type continuous gravity concentrator.
- Regrind area crane, 20t capacity, 36m bridge span.
- Associated pumps.

17.3.3.3 Cleaner and Cleaner-Scavenger Magnetic Separation

The first regrind cyclone overflow will gravity flow to a feed distributor ahead of eight cleaner LIMS units, with the tailings from each unit gravity flowing to a dedicated cleaner-scavenger LIMS to recover any remaining magnetic minerals. The combined concentrate from all cleaner and cleaner-scavenger LIMS will be sent to the second regrind stage and the non-magnetic tailings to the combined tailings pumbox for discharge in the tailings facility. The primary cyclone area crane will also be used for the cleaner and cleaner-scavenger LIMS.

The area will include the following major equipment and facilities for the initial operation stage:

- Eight (8) cleaner LIMS, single roll of 1.2 m diameter by 4 m width, magnetic field strength of 1,800 gauss.
- Eight (8) cleaner scavenger LIMS, single roll of 1.2 m diameter by 4 m width, magnetic field strength of 1,800 gauss.
- Associated feed distributors and pumps.

17.3.3.4 Second Regrind

The combined magnetic concentrate from all cleaner and cleaner-scavenger LIMS will be sent to the second regrind mill feed pump box, and then pumped to the stirred mill for regrinding, where high specific gravity (SG) ceramic grinding media of 3-5 mm diameter will be used to achieve the final regrind size with a P_{80} of 18 µm. The ceramic media is transferred from the media bin to the mill feed pump box where it is mixed with the feed slurry and pumped into the mill. The second regrind circuit is designed to have a direct-feed arrangement, with mill discharge reporting directly to the recleaner magnetic separation circuit.

The area will include the following major equipment and facilities:

- One stirred regrind mill with LRS and 5,000 kW motor.
- Associated pumps.

17.3.3.5 Recleaner and Recleaner-Scavenger Magnetic Separation

The second regrind mill discharge will be discharged into a pump box and pumped to a distributor to feed a set of six 3-stage recleaner LIMS, where the magnetic concentrate from the one stage will be fed to the next stage for further cleaning. The tailings from each unit will gravity flow to a recleaner scavenger LIMS to recover any remaining magnetic minerals. The magnetic concentrate from all units will be combined and sent for thickening ahead of the awaruite flotation circuit.

The primary cyclone area crane will also be used for the recleaner and recleaner-scavenger LIMS maintenance.

The area includes the following major equipment and facilities:

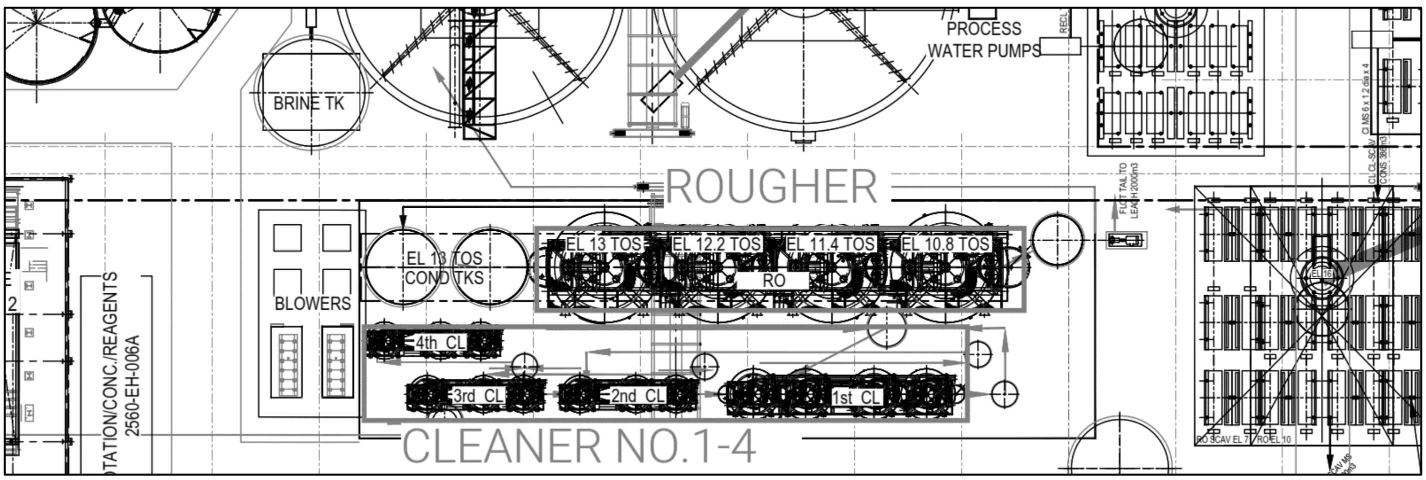
- Six, 3-stage recleaner LIMS triple roller in series, three rolls each of 1.2 m diameter by 4 m width with a magnetic field strength of 1,800 gauss.

- Six recleaner-scavenger LIMS, single roll of 1.2 m diameter by 4 m width with a magnetic field strength of 1,800 gauss.
- Associated feed distributors and pumps.

17.3.4 Awaruite Flotation

The awaruite flotation circuit is designed for an operating time of 8,060 hours a year, equivalent to 92% availability. The awaruite flotation circuit will include roughing and four-stage cleaning flotation separation at mildly acidic pH levels. The circuit will produce the final flotation awaruite concentrate containing 60% Ni and 25% Fe. The awaruite flotation area is depicted in Figure 17-6 and outlined in further detail in the following subsections.

Figure 17-6: Flotation Area Layout



Source: Ausenco, 2023.

17.3.4.1 Flotation Feed Thickening

The final magnetic concentrate from the recleaner magnetic separation circuit will be sent to a high-rate thickener to increase the pulp solids density to 70% by weight in order to minimize the addition of process water input to the flotation circuit. Thickened magnetic concentrate from the flotation feed thickener will go to conditioning where it is diluted with recycled flotation/leach brine. Thickener overflow will gravity flow to the process water tank. Spillage in the flotation feed thickener area will be pumped to the thickener feed box.

The thickener area flotation crane will be used for maintenance for the flotation feed thickener and pre-leach thickener.

The area will include the following major equipment and facilities:

- One flotation feed thickener, 22 m diameter.
- One thickener area crane, 10 t capacity, 28 m bridge span.

- Associated pumps.

17.3.4.2 Conditioning and Rougher Flotation

Thickened flotation feed slurry will be pumped to the first of two flotation conditioning tanks. In the first conditioning tank, recycled brine and sulphuric acid will be added to dilute the flotation feed to a pulp solids density of 25% by weight, and to decrease the pH to 4.0 – 4.5, respectively. Over the course of flotation (conditioning, roughing, cleaning) around 2.2% of the nickel in the magnetic concentrate is estimated to dissolve into solution, along with other elements, primarily magnesium, and iron. Collector sodium isopropyl xanthate (SIPX) will be added in the second conditioning tank to activate awaruite.

Conditioned feed slurry will be fed to a series of four rougher flotation cells along with the first flotation cleaner tailings stream. The rougher flotation concentrate will be pumped to the cleaner flotation circuit. Flotation concentrate launder wash water will be supplied from the brine tank to minimize circuit dilution. The rougher flotation tailings will be directed to the pre-leach tailings thickener.

SIPX and a glycol frother will be dosed in the rougher and first cleaner cells. A low-pressure blower will provide air to the flotation cells. Spillage in the rougher flotation area will be pumped to the first rougher cell. All cells in the awaruite flotation circuit will be closed and ventilated via a centralized air ventilation system.

The awaruite flotation area overhead crane will be used for maintenance on the rougher and cleaner flotation area, as well as the reagent area.

The area will include the following major equipment and facilities:

- Two agitated rougher conditioning tanks, 50 m^3 each.
- Four rougher flotation cells, 160 m^3 each.
- One flotation area crane, 20 t capacity, 40 m bridge span.
- Associated pumps, blower, and off-gas ventilation systems.

17.3.4.3 Cleaner Flotation

The cleaner flotation circuit is a four-stage closed flotation process. The first cleaner concentrate will be pumped to the next cleaner flotation stage until the fourth cleaner flotation stage, where the final awaruite concentrate will be collected and pumped to dewatering. Flotation concentrate launder wash water will be drawn from the brine tank. The intermediate tailings of each of the cleaners will flow by gravity back to the previous flotation stage for reprocessing.

Collector and frother will be dosed into the cleaner flotation circuit along with sulphuric acid to maintain a pH range of 4.5 – 4.8. Spillage in the cleaner flotation area will be pumped back to the first cleaner cells. All cells in the awaruite flotation circuit will be closed and ventilated via a centralized air ventilation system.

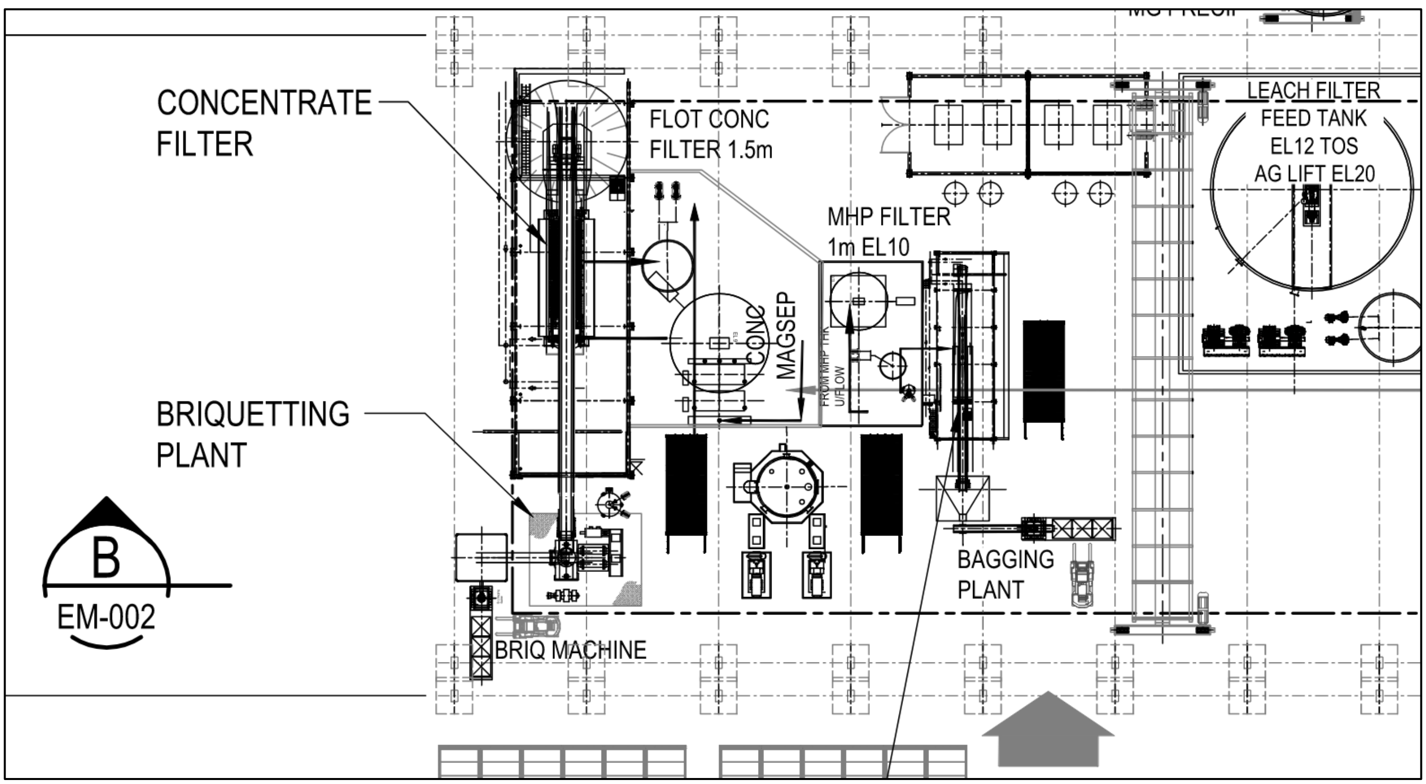
The area will include the following major equipment and facilities:

- Four first cleaner flotation cells, 10 m^3 each.
- Three second cleaner flotation cells, 5 m^3 each.
- Three third cleaner flotation cells, 3 m^3 each.
- Three fourth cleaner flotation cells, 3 m^3 each.
- Associated pumps, blower, and air ventilation systems.

17.3.5 Awaruite Concentrate Dewatering and Loadout

The awaruite concentrate will be dewatered first by a LIMS, then followed by filter press prior to briquetting for packaging. The dewatering and loadout area is depicted in Figure 17-7 and outlined in further detail in the following subsections.

Figure 17-7: Concentrate Dewatering and Loadout Area Layout



Source: Ausenco, 2023.

The concentrate dewatering circuit will consist of a LIMS system and concentrate filter press to dewater the awaruite concentrate prior to briquetting, loadout, and shipment. The awaruite concentrate from the cleaner flotation circuit will be pumped into a feed box that will gravity feed the dewatering LIMS, which will increase the pulp solids density from

20% to 65% solids by weight. The partially dewatered concentrate will then pass through a filter press to further decrease the water content to 86% solids by weight. The filter cake will be washed with fresh water to displace brine.

The brine removed from the concentrate by the dewatering LIMS and filter press will be partially recirculated to the dewatering LIMS, while the rest will be pumped to the brine tank.

The filtered awaruite concentrate will be discharged to a reversible conveyor that can convey the concentrate to either the briquetting plant, or an emergency stockpile when the briquetting plant is under maintenance. The awaruite concentrate will be compressed into briquettes as the final saleable product using an ambient temperature, binder-less roller press. The produced briquettes will be packaged in 1 m^3 United Nations (UN)-rated bags and stored in the concentrate storage before being loaded into shipping containers for shipment off-site.

The filtration area overhead crane will be used for maintenance on the awaruite concentrate filter press, as well as for the leach filter and MHP filter.

There has been a small allowance for concentrate inside the process facility prior to being loaded into bags and containers. The platform outside the bagging area will have sufficient area for storing and handling double-stacked container storage equivalent to 3 days process demand for managing containers and placing on to trucks.

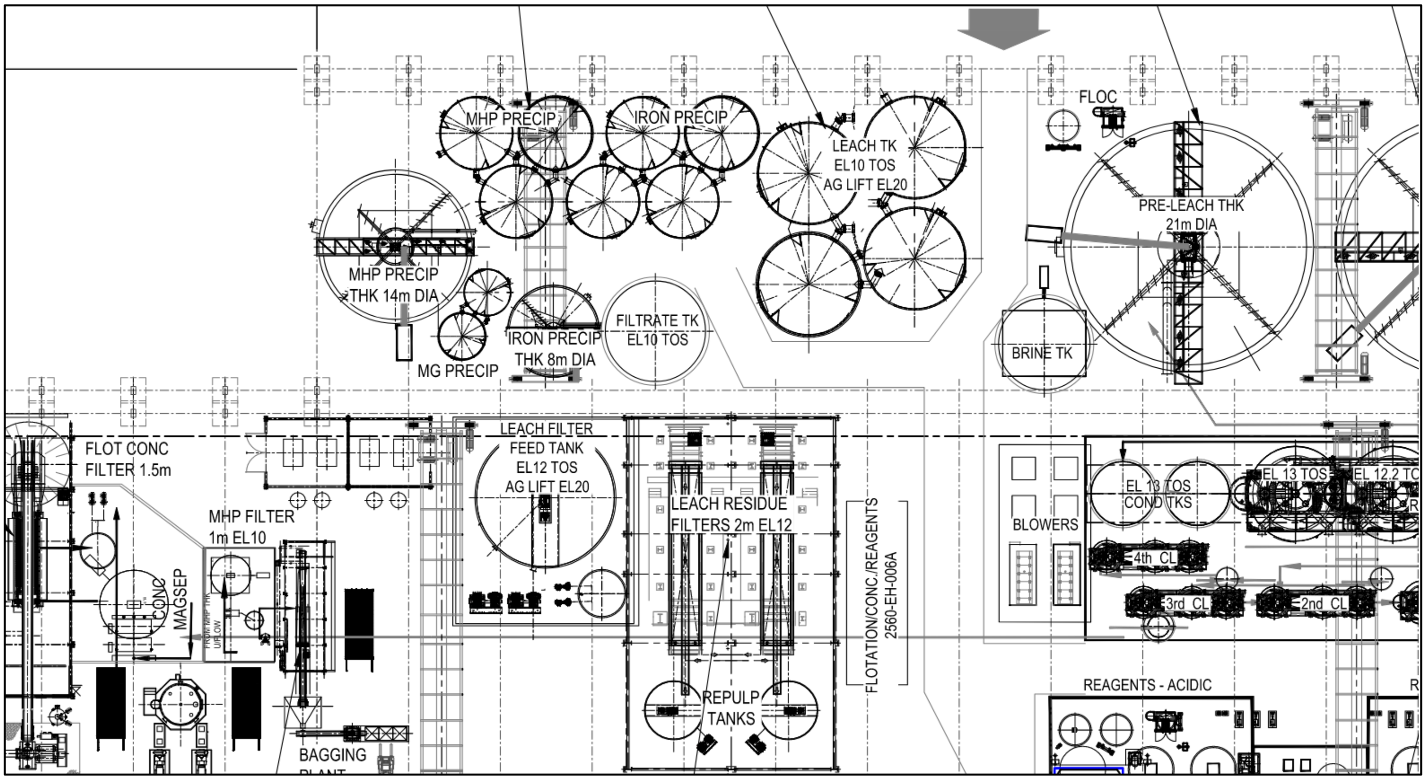
The area will include the following major equipment and facilities:

- One single roll dewatering LIMS, single roll 0.9 m diameter x 0.6 m width with a magnetic field strength of 1,200 gauss.
- One awaruite concentrate filter press, 68 m^2 total filtration area.
- Awaruite briquetting machine, 521 mm diameter roll.
- One filter area crane, 25 t capacity, 40 m bridge span.
- Associated conveyors and feeders.

17.3.6 Flotation Tailings Leaching

The flotation tailings will be leached to recover acid soluble nickel which, when combined with leaching that naturally occurs during flotation, increase overall DTR Ni recovery by a further 6.5%. The flotation tailings leaching area is depicted in Figure 17-8 and outlined in further detail in the following subsections.

Figure 17-8: Flotation Tailings Leaching Area Layout



Source: Ausenco, 2023.

The flotation tailings will be thickened in a high-rate thickener to increase the pulp solids density to 50% by weight and decrease required leach tank volume. Overflow from the thickener will be directed to the brine tank with the underflow pumped to four agitated leaching tanks. Leaching will be accomplished by adding sulphuric acid and sparging air to the agitated leach tanks. Approximately 50% of the nickel in the leach feed will be extracted into solution after a residence time of 8 hours.

The leach residue will be pumped to the leach filter feed tank where it will be joined by a relatively small flow of iron precipitate thickener underflow. The leach filter feed tank will have 8 hours surge capacity. The combined slurries will be pumped to a filter press and the filter cake will be washed with process water to recovered dissolved nickel. The filtrate and wash from the filter press will be pumped to the brine tank. The leach residue cake will be conveyed to the repulping tank before being pumped to the combined tailings pump box then discharged to the tailings facility.

The area will include the following major equipment and facilities:

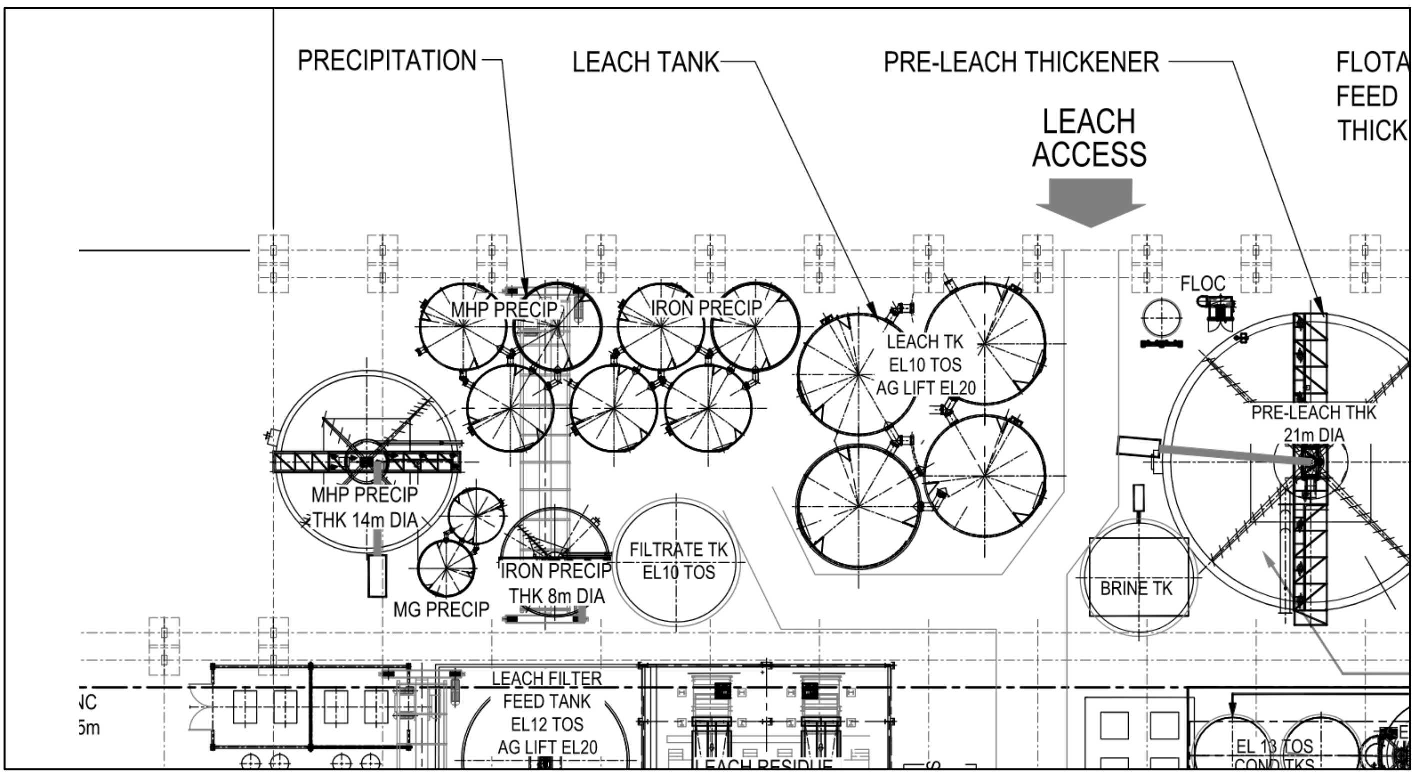
- One pre-leach thickener, 21 m diameter.
- Four agitated leaching tanks, 572 m^3 each.
- Leach filter feed tank, 2400 m^3.

- Two leach filter presses, 992 m^2 total filtration area per filter.
- Associated pumps and solution tanks.

17.3.7 Solution Purification and MHP Precipitation

A bleed stream of brine from awaruite flotation and tailings leach residue dewatering areas will be sent for impurity removal and MHP precipitation. The solution purification and MHP precipitation area is depicted in Figure 17-9 and outlined in further detail in the following subsections. The MHP bagging circuit will be located adjacent to the awaruite concentrate briquetting plant (Figure 17-7).

Figure 17-9: Solution Purification and MHP Precipitation Area Layout



Source: Ausenco, 2023.

Excess brine from the brine tank will be pumped to the three-stage precipitation circuit for impurity removal and MHP production.

The first precipitation stage will remove dissolved iron by raising the pH to 5.5 in four aerated and agitated tanks. Limestone will be added to the first three tanks and lime will be added to the fourth tank. Residence time for each tank will be one hour. The last iron precipitation tank discharge will be pumped to the iron thickener where a portion of the underflow will be recycled back to the reactors for precipitate seeding and the balance will be pumped to the leach residue filtration circuit. The overflow of the iron thickener will gravity flow to the second precipitation stage to precipitate MHP.

The second precipitation stage will increase the pH to 8.1 by adding caustic calcined magnesia in three agitated precipitation tanks, with a retention time of 1 hour per tank. The last MHP precipitation tank discharge will be thickened and filtered prior to packaging. Most of the thickener underflow will be recycled back to the reactors to precipitate seeding while that balance will advance to filtration. Overflow from the thickener will be gravity flow to the third precipitation stage to precipitate magnesium. MHP filter filtrate will be recirculated to the thickener. The filtered MHP product will have a moisture content of 50% by weight and will be packaged into 1 m^3 UN-rated bulk bags with liner and stored in the concentrate storage before being loaded into shipping containers, which will then be loaded onto trucks for shipment off-site.

The final precipitation stage will precipitate magnesium from the MHP thickener overflow solution. Magnesium precipitation will take place in two agitated tanks using lime. Residence time will be 15 minutes per tank. The slurry will be pumped to the leached residue repulp tank and ultimately the tailings facility.

The precipitation area overhead crane will be used for maintenance on the leach circuit, as well as for the solution purification and MHP precipitation areas.

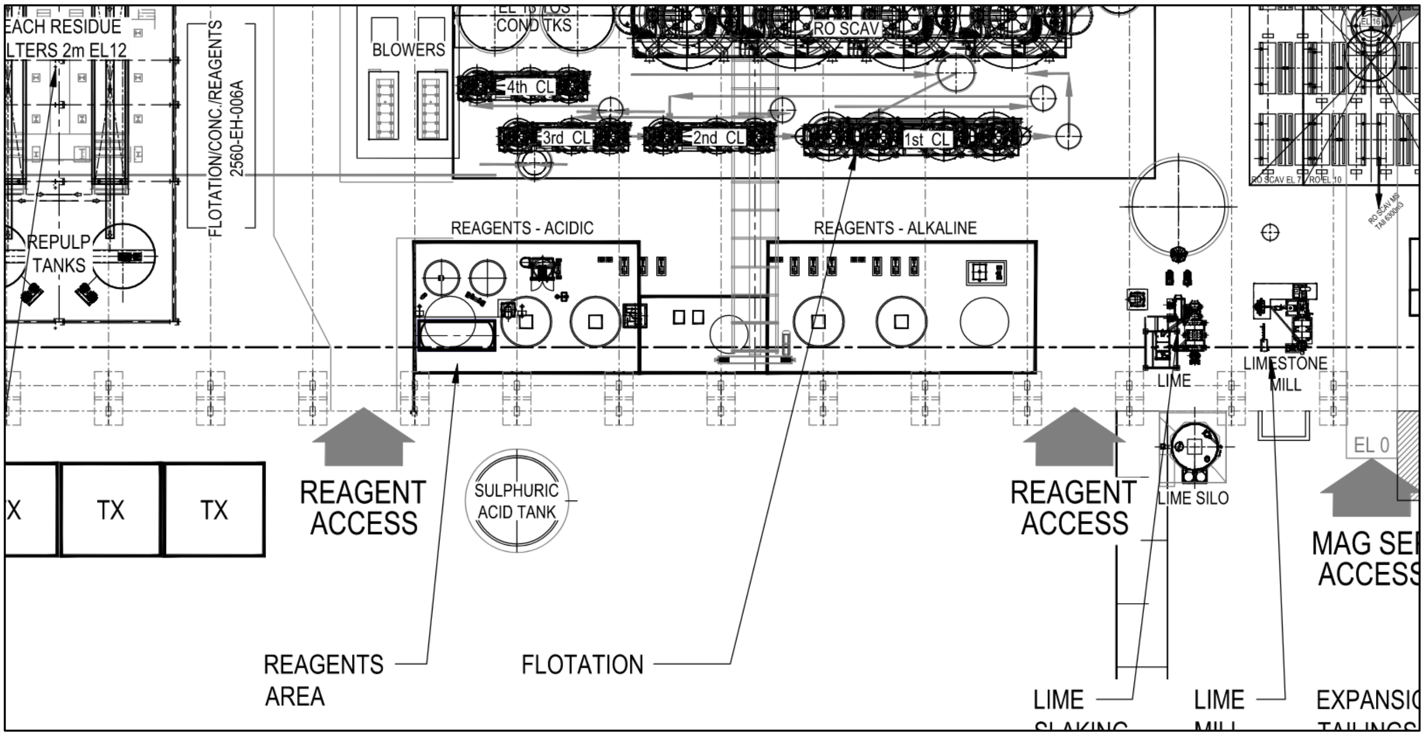
The area will include the following major equipment and facilities:

- Four iron precipitation tanks, 233 m^3 each.
- One iron precipitation thickener, 8 m diameter.
- Three MHP precipitation tanks, 233 m^3 each.
- One MHP precipitation thickener, 14 m diameter.
- One MHP filter press, 52 m^2 total filtration area.
- Two magnesium precipitation tanks, 62 m^3 each.
- One precipitation area overhead crane, 10 t capacity, 28 m bridge span.
- Associated pumps and tanks.

17.3.8 Reagents and Grinding Media

The reagent preparation area will be located at the south side of the process plant (Figure 17-10). Annual reagents consumption is shown in Table 17-2. Reagents required for the project are described below. The reagent area storage is located adjacent to the awaruite flotation cells and will share the same overhead crane. As a safety precaution, acidic and basic reagents will be stored separately on each side of the reagent area.

Figure 17-10: Reagent Area Layout



Source: Ausenco, 2023.

Table 17-2: Reagents Consumption

Reagent	Phase 1 Consumption (t/a)	Phase 2 Consumption (t/a)
Sulphuric Acid	75,600	101,000
Collector (SIPX)	570	760
Glycol Frother	1,195	1,600
Flocculant	75	100
Limestone	475	635
Lime	38,490	51,400
Magnesia	2,150	2,970

17.3.8.1 Sulphuric Acid (H_2SO_4)

Sulphuric acid will be received on site as a bulk liquid with a minimum purity of 93% H_2SO_4. Sulphuric acid will be stored in a holding tank outside the south side of the mill with 48-hour consumption capacity and pumped to the awaruite flotation circuit and tailings leaching for pH control. The sulphuric acid holding tank will be climate controlled to prevent freezing given the climatic conditions of the region.

17.3.8.2 Collector (SIPX)

SIPX collector will be received on site in 1.1 t super sacks in pellet form and dissolved into a mixing tank. The diluted collector mixture will then be pumped to a day tank from where it will be dosed to the rougher conditioning tanks and first cleaner cells.

17.3.8.3 Frother

Glycol frother will be received on site in the form of intermediate bulk containers (IBC) with a bulk weight of 800 kg. Frother will be stored in a storage tank before being pumped for use in the rougher and cleaner flotation.

17.3.8.4 Flocculant

A mixing and storage system will prepare flocculant to facilitate thickening processes. An anionic polyacrylamide flocculant will be received on site in 750 kg bags containing dry powder with a minimum purity of 99%. Individual flocculant bags will be hoisted over and emptied into a hopper. A flocculant screw feeder and wetting head will mix the powdered flocculant and fresh water in a mix tank before pumping to a storage tank.

17.3.8.5 Limestone ($CaCO_3$)

Limestone will be received on site in supersacks and fed to the limestone ball mill via a feed conveyor. After grinding, the limestone slurry will be pumped to a cyclone for classification. Cyclone overflow will go through a trash screen and gravity flow into the limestone slurry tank with any trash reporting to a bin. The cyclone underflow will be returned to the ball mill. The stored limestone slurry will be circulated in a ring main and sent to the first three iron precipitation cells for pH control.

17.3.8.6 Lime (CaO)

Lime will be received on site via lime tankers. The lime will be blown from the tanker into the lime silo. Lime will be drawn from the silo with a screw feeder and will be passed over the lime screen with the undersize gravity flowing to the detention lime slaker. The slaker will be equipped with a scrubber and fan system to ventilate the heat and gases generated by the slaking process. The slaked lime will be transferred to a storage tank and distributed in a ring main from which the lime slurry will be dosed to the fourth iron precipitation cell and the two magnesium precipitation cells. A scrubber will be installed on the area ventilation.

17.3.8.7 Caustic Calcined Magnesia (MgO)

Magnesia will be received on site in bulk bags and be loaded into a day silo. A venturi contactor, be fed from the silo via a screw feeder, will be used to mix the solid magnesia with fresh water immediately before use. The silo will be installed close to the MHP precipitation tank to minimize time between mixing and use.

17.3.8.8 Grinding Media

The grinding media including forged steel balls in different sizes and high density ceramic media will be delivered to site in bulk. The steel balls will be unloaded to the respective storage bin and loaded into the SAG mill, primary ball mills, and the first regrind ball mill by kibble. The ceramic media will be transferred to the media hopper and pumped to the second regrind mill.

17.3.9 Services and Utilities

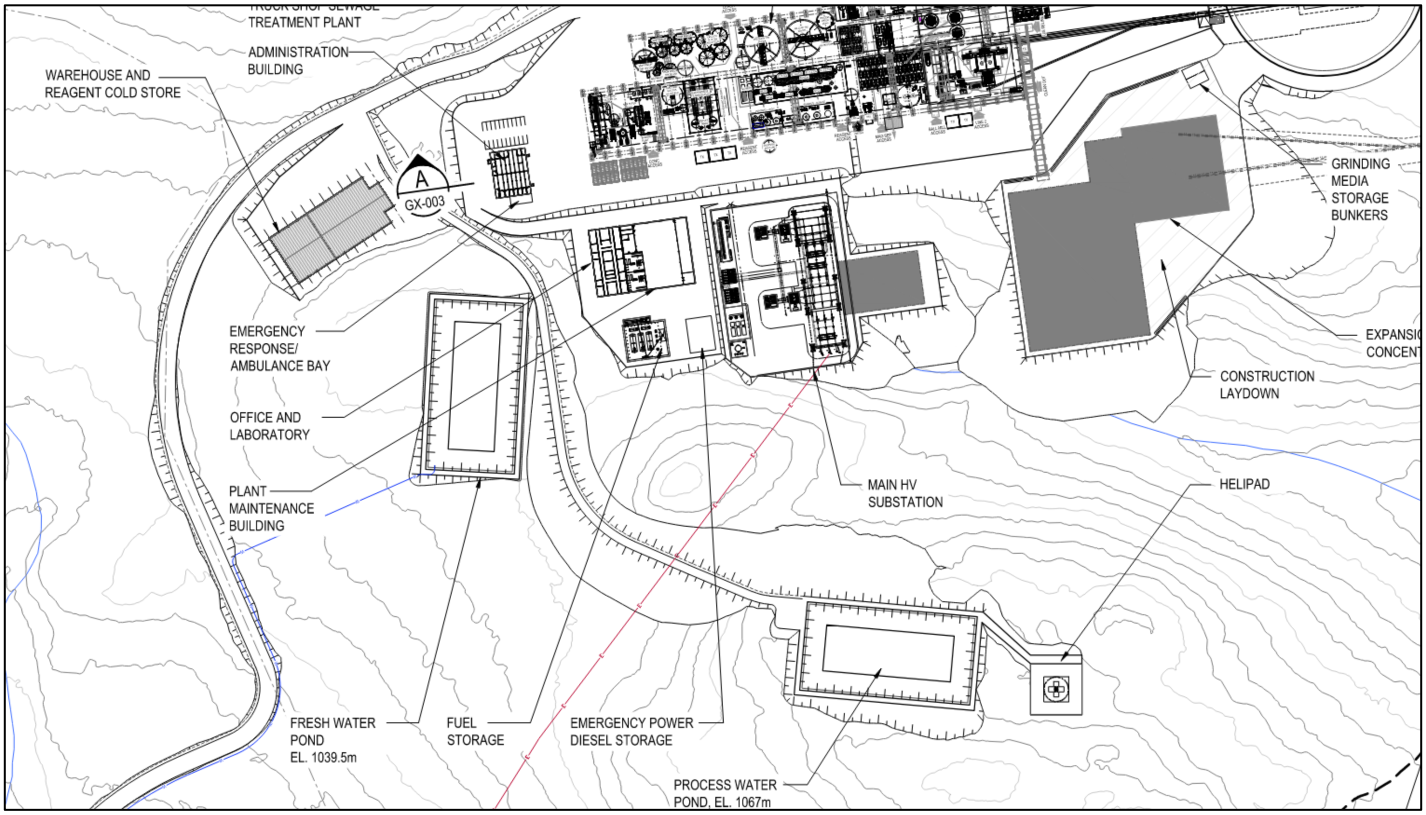
The fresh water pond and process water pond will be located southwest of the processing plant and main high voltage (HV) substation, within minor earthwork quantity areas at elevations that make use of local topographic high points in relation to the plant facilities such that gravity flow can be utilized. Both ponds are sufficiently sized to handle the volumetric requirements for Phase 2.

The fresh water storage tank is located in the mill and is designed to be sufficient for the increased mill throughput in Phase 2. An additional gland water pump and fresh water pump will be required in Phase 2 to maintain the required pressure and volumetric flow to the end users.

The potable water treatment plant and potable water storage tank are located in the mill and are also designed to be sufficient for the mill expansion. An additional potable water pump will be required to distribute the water to all onsite users in Phase 2.

Figure 17-11 shows the general arrangement and details are outlined below in the following subsections. Other onsite facilities are covered in detail in Section 18.

Figure 17-11: Services and Utilities Area Layout



Source: Ausenco, 2023.

17.3.9.1 Power

The main high voltage substation is located just south of the mill and will receive power from a 230 kV overhead transmission line from BC Hydro's Glenannan substation approximately 155 km away. Phase 1 will utilize one major electrical room with a second one installed when Phase 2 begins. Power will be stepped down to 34.5 kV and connected to different consumers through overhead transmission lines. Depending on the end user, power will be further stepped down to 4.16 kV, 600 V, and 120/207 V to accommodate medium and low voltage switchgear.

The total installed power for the process plant and estimated power consumption for each phase is provided in Table 17-3. Site power draw is further discussed in Section 18.9.2.

Table 17-3: Power Requirements and Energy Consumption

Phase	Installed Power (MW)	Operating Power (MW)	Energy Consumption (MWh/a)
Phase 1	168	145	1,070,664
Phase 2	254	219	1,536,002

17.3.9.2 Process Water

Reclaim water from the tailings facility supernatant pond will be pumped via multiple pump barges and overland pipelines to the process water pond. The process water pond will supply at least a 1-hour process water flow rate. Makeup water will be provided by the fresh water storage pond.

The process water pond will be elevated relative to the mill which will allow the gravity flow of water via overland pipes to the primary grinding circuit, the primary process water consumer in the concentrator. Process water for other uses in the concentrator will be first routed to the process water tank inside the mill, then pumped to the various consumers throughout the plant site.

The area will include the following major equipment and facilities:

- One process water tank, 460 m^3 live volume.
- One process water pond, 15,000 m^3 live volume.
- Associated pumps.

17.3.9.3 Fresh Water

Fresh water will be pumped utilizing vertical turbine pumps from Trembleur Lake via a 14 km overland pipeline to a fresh water pond. The fresh water pond will be located at an elevation which allows gravity flow to a fresh/fire water tank. Fresh water is pumped for use in processing and as gland water and for process makeup water as necessary. The fresh water pond is sized to provide a minimum of 10-hour residence time plus capacity for the fire water reserve and essential water consumption (excludes process makeup water). Horizontal centrifugal pumps in a duty/standby arrangement will distribute fresh water to the various consumers throughout the plant site.

The fresh water intake system is designed for instantaneous flows 600 m^3/h in Phase 1, and 900 m^3/h in Phase 2.

The area will include the following major equipment and facilities:

- One fresh water tank, 1,400 m^3 live volume.
- One fresh water pond, 15,000 m^3 live volume.
- Associated pumps.

17.3.9.4 Fire Water

Fire water for the process plant will be sourced from the fresh water tank. A dedicated pump skid consisting of an electrical pump, jockey pump, and diesel pump that will draw water from the dedicated fire water reserve volume into a fire water reticulation system that services the plant site areas. The minimum required volume at the bottom of the fresh water tank will be maintained for use by the fire water system.

17.3.9.5 Potable Water

Potable water will be sourced from the fresh water tank, treated in a potable water treatment plant, and stored in a storage tank. The potable water plant is designed to supply water to the mine and mill.

17.3.9.6 Gland Seal Water

Gland seal water will be sourced from the fresh water tank, passed through filters to remove particulate, and pumped to various users throughout the plant site area via a distribution system.

17.3.9.7 Air Services

17.3.9.7.1 Process Air

Awaruite flotation air blowers will provide low pressure air to the rougher and cleaner flotation cells. The blowers will generate air at the highest pressure required by the flotation cells, and pressure reducers will step-down the pressure for any flotation cells requiring lower pressures. Two blowers will be installed to meet flotation air requirements for Phase 1.

Separate air blowers for the leach circuit will provide lower pressure air to the leaching tanks and iron precipitation tanks.

17.3.9.7.2 Plant and Instrumentation Air

Rotary screw air compressors will supply air to various users throughout the plant site via air distribution system, and an air dryer will provide instrument air as required.

The awaruite concentrate filter, leach residue, and MHP filters will have a dedicated compressor to service the core blow, membrane squeeze, and air dry requirements.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The Baptiste Nickel Project will include the following facilities:

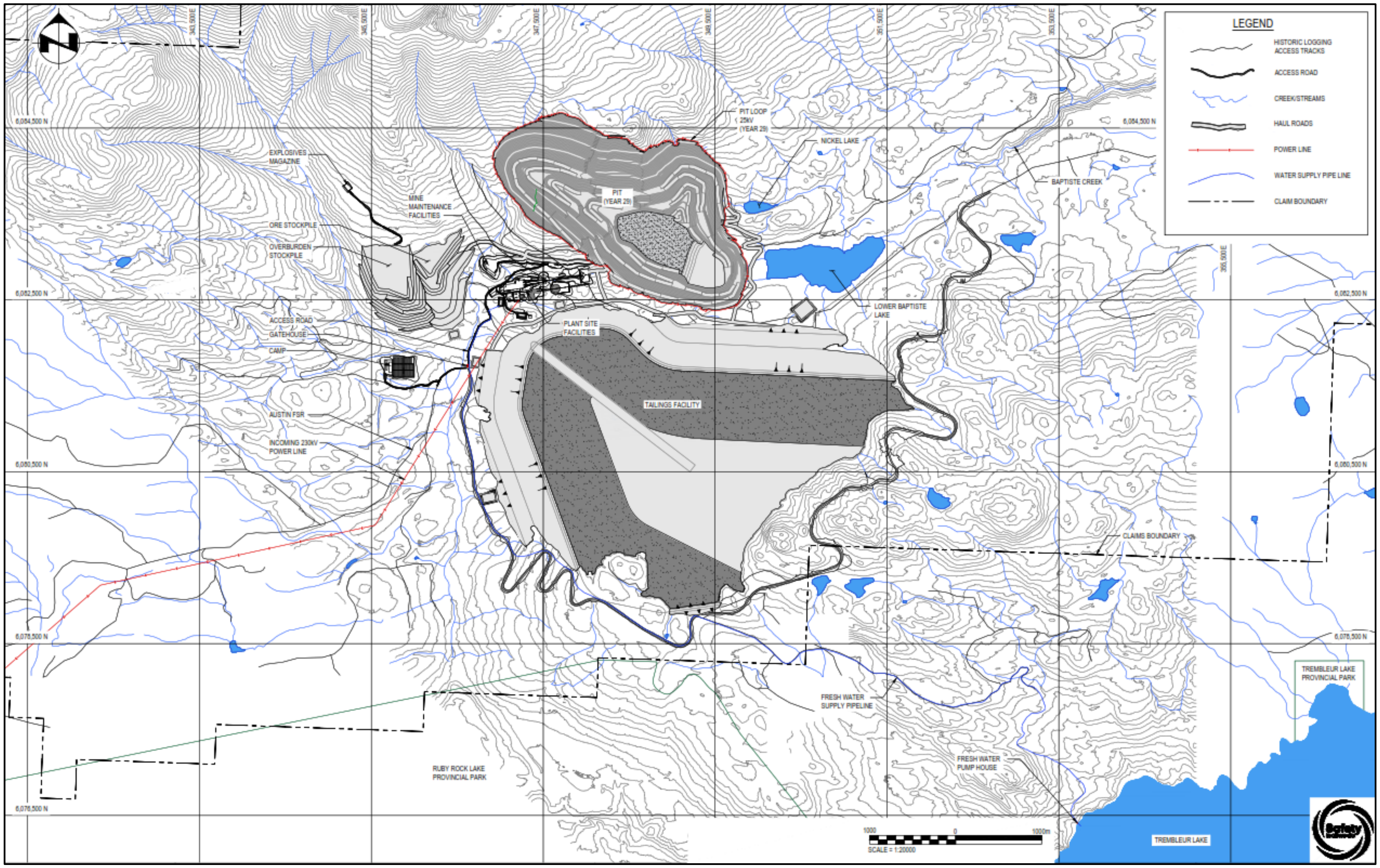
- Mining facilities including the mine office and dry, truck shop, explosives storage and batching facility, diesel fuel storage and distribution, operational ore stockpile, mine electrical loop, mine dewatering, tire change facility, and truck wash.
- Processing facilities including the primary crushing and ROM pad, coarse ore stockpile and reclaim conveyors, process plant, offices and laboratory, and plant maintenance building.
- Waste and water management infrastructure including the tailings facility, tailings distribution system, water reclaim system, downstream water management ponds, and overburden stockpile.
- General facilities including the camp, gatehouse, emergency response/ambulance bay, warehouse, reagent cold storage, administration building, communications, emergency power diesel storage, potable fresh, process and potable water storage and distribution systems, main HV substation, site sewage systems, and solid waste treatment.

The locations of the site facilities were based on the following criteria:

- Locate the facilities within the claim boundaries, to the greatest extent possible.
- Utilize topographic advantages for all facilities to optimize construction costs.
- Locate the mine, process plant, and tailings facility within a single catchment area to reduce the mine footprint area, to the greatest extent possible.
- Locate the primary crushing facilities adjacent and with a similar elevation to the open pit exit ramp to optimize hauling costs as much as possible.
- Locate fresh and process water ponds close to the plant site area at a sufficient elevation to maximize gravity distribution options.
- Maximize the use of existing local infrastructure where possible, including the existing FSR network and connection to the BC Hydro electrical grid.
- The tailings facility location was selected based on currently available technical and environmental criteria. Early engagement with local communities and Indigenous Rightsholders is ongoing and will continue to inform tailings facility siting assessments.

The overall site layout is shown in Figure 18-1.

Figure 18-1: Overall Site Layout



Source: Ausenco, 2023.

18.2 Site Access

18.2.1 Off-site Access Roads

The project site can be accessed from Fort St. James by using an existing 170 km long road network made up of both paved public roads and gravel FSRs. Currently the off-site access road includes 40 km of the paved Tachie Road maintained by the Ministry of Transportation and Infrastructure, 40 km of Leo Creek FSR, 49 km of Kazcheck FSR, and 41 km of Austin FSR. The current road network sufficiently supports current project development activities and is foreseen to continue doing so until the completion of the Early Works program that precedes the formal construction period.

Presently nine bridge crossings and ten major culvert crossings (diameters greater than 1,200 mm) are part of the existing road network. Existing crossing structures permit the use of BCL-625 design vehicles (63,710 kg G.V.W.) with highway legal axle loads. The overall FSR road network is limited to a BCFS L75 Design Vehicle (68,040 kg G.V.W.) due to the current load rating of two existing structures. However, only one L75 structure will be used for project access following the construction of the Middle River Bypass.

To reduce the overall length of the off-site access road network, a Middle River bypass is proposed. Once completed, this will bypass approximately 56 km of the existing FSR network, including the existing Middle River bridge. Implementing the Middle River Bypass will require a new 7 km resource road segment construction and a new 253 m, overall length, steel girder with composite concrete deck bridge across Middle River. The new Middle River Bridge will consist of six 30 m jump-spans and one 70 m main-span founded on driven pile abutments and piers. The Middle River Bypass will also require a 12m concrete slab girder bridge on pre-cast spread footings, and a 24 m steel girder with composite concrete deck bridge on driven piles. Implementing the Middle River Bypass will reduce the number of existing crossings used for the Project access by fourteen.

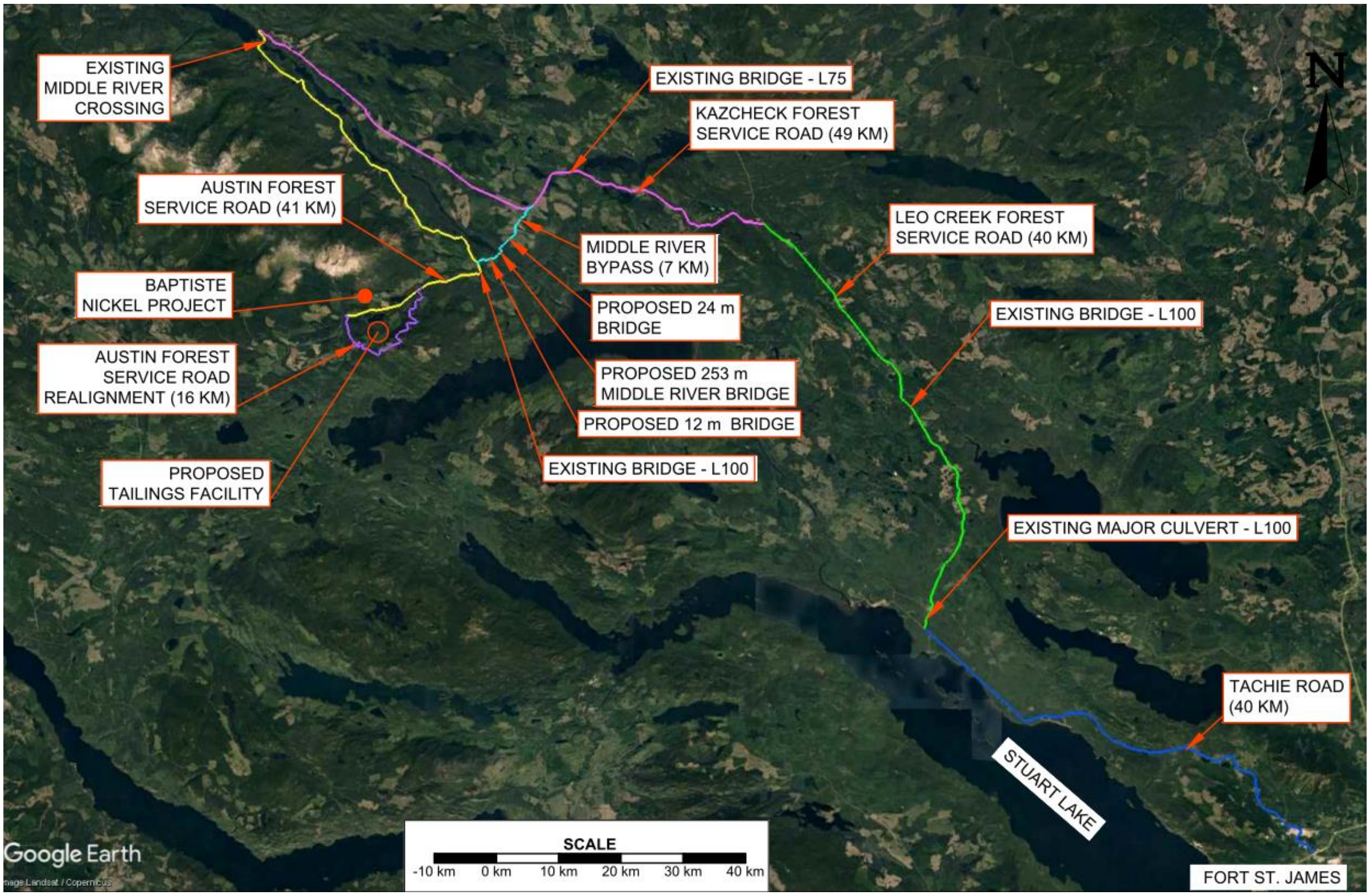
Prior to construction of the tailings facility, 6 km of the existing Austin FSR must be realigned around this facility. The Austin FSR realignment will require 16 km of new road construction through sections of steep ground and nine 2,400 mm culvert stream crossings. Implementing the Austin FSR realignment will reduce the use of one existing stream crossing on the existing Austin FSR.

Sections of the existing FSRs that would be used for project access following the construction of the Middle River Bypass and the Austin FSR realignment will require upgrades to increase road strength and mitigate seasonal axle load restrictions to facilitate year-round hauling. Road structure upgrades proposed will include the application of moisture wicking geosynthetics and high fines surfacing aggregate along 68 km of existing FSRs. The application of geosynthetic and high fines surfacing aggregate would also be applied along the proposed Middle River Bypass and Austin FSR realignment.

Following the implementation of the Middle River bypass and Austin FSR realignment, the overall length of the off-site access road from Fort St. James to the Project site will be approximately 131 km, versus the current 170 km length. This will reduce the one-way commute time by approximately 39 mins (from 150 to 111 mins).

In addition to proposing an allowance that would upgrade existing FSRs to facilitate year-round hauling, Onsite Engineering Ltd. prepared conceptual road designs for the Middle River bypass and Austin FSR realignments to assist in estimating their associated capital costs. A summary of the off-site access route is shown in Figure 18-2 below.

Figure 18-2: Off-site Access Road System Overview



Source: Onsite Engineering, 2023

18.2.2 Mine Roads

A series of mine roads will allow the haulage of ore and waste rock from the open pit to the scheduled destinations and from the operational ore stockpile to the primary crusher. The mine roads were designed with the following key considerations:

- Ramp width of 40 m to allow for two-lane traffic;
- Ramps with a maximum grade of 10% (8% desired); and
- Designed to accommodate a fleet of 300 t class haul trucks.

18.2.3 On-site Plant Roads

The plant site roads will be 7 m wide, gravel surfaced, uniformly graded, free draining and compacted to meet applicable standards. The process plant pad and road pavement thickness, materials, surface, etc. will be made to withstand the expected maximum axle loads and traffic density.

The maintenance roads will be designed as 5 m wide, one-lane roadways.

18.2.4 Railway

Canadian National (CN) Rail operates a railway from Fort St James through Prince George, Vanderhoof, Smithers, and Terrace to the Port of Prince Rupert. The primary mode of transport between Fort St. James and the project site will be trucking via existing highway and upgraded forestry service roads.

An out-of-service rail line, also owned by CN Rail, follows the east bank of Middle River, and runs through the northeastern margin of the property. It is not envisioned that this line will be refurbished for project use.

18.2.5 Airports

The closest international airport to the Project is the Prince George Airport (YXS), approximately 170 km east of Fort St. James. Fly-in-fly-out (FIFO) personnel will arrive and depart from YXS. Workers flying in will transfer onto buses to travel to site.

18.3 Mine Maintenance Facilities

The mine maintenance facilities are located within a dedicated pad adjacent to (but at a higher elevation than) the process plant area, see Figure 18-3. The truck shop includes four bays for mining and auxiliary equipment maintenance (with two additional bays added in Year 6). Other mine maintenance facilities include a truck wash, and tire change in addition to a mine dry, office, and mechanical shop. The mine dry comprises a single room split into two areas for the purpose of keeping miners' work clothes separate from their regular attire. See Table 18-1 for a breakdown of the mine maintenance facilities. Both the truck wash and tire change are their own detached structures, with the other facilities listed as areas within the mine maintenance building (see Figure 18-3).

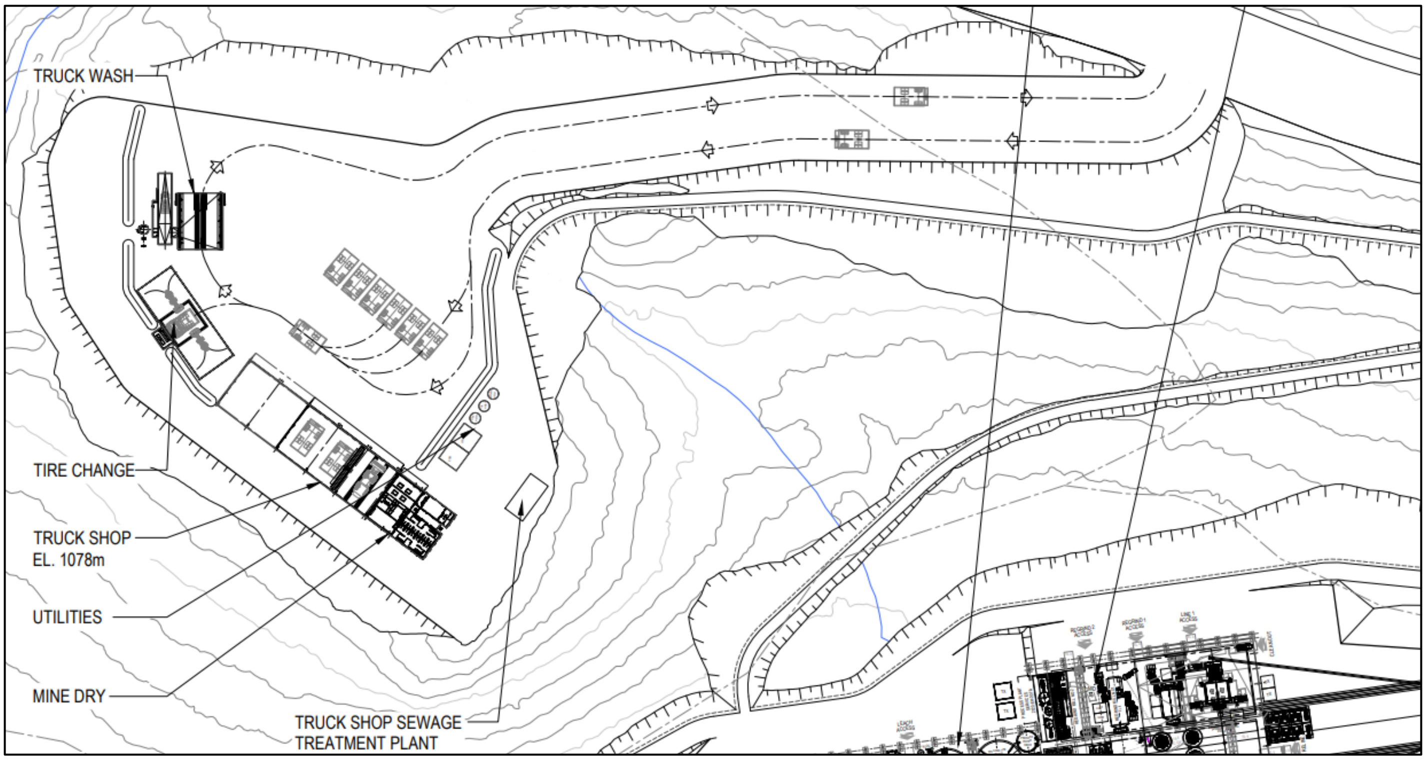
Table 18-1: Mine Maintenance Facilities

Facility Name	Construction Type	Length (m)	Width (m)	Height (m)	Area (m^2)
Tire Change	Pre-Engineered	16	14	12	224
Mine Dry*	Pre-Engineered	43	25.5	8	1,101
Truck Shop*	Pre-Engineered	52	25.5	14	1,326
Truck Wash	Modular-Fabricated	26	18.6	12	484
Mine Mechanical Shop*	Pre-Engineered	25.5	17	3.5	434
Mine Office	Modular	17	7.5	3.5	128

* These are areas within the single mine services building.

An explosives storage facility will be located north of the operational ore and overburden stockpiles, however while at this stage of the study it has not been sized, an allowance has been made in the capital cost estimate. Similarly, a fuel station facility has been allowed for in addition to four 100,000 L capacity diesel storage tanks in the initial phase and three additional tanks in the expansion phase. The facility itself has not been engineered at this stage of the study. Further on-site mining infrastructure includes the open-pit dewatering system (see Section 18.11.2) and a 25 kV power loop.

Figure 18-3: Mine Maintenance Facilities



Source: Ausenco, 2023.

18.4 Process Infrastructure

The process plant will be composed of several pre-engineered buildings that house the main process circuits within the plant building. The primary crusher will be located approximately 600 m east from the concentrator building. Table 18-2 lists the buildings located within the main process plant. All pre-engineered buildings noted below represent the footprint of individual areas within the single concentrator building.

Table 18-2: Process Infrastructure Buildings

Building/ Area Name	Construction Type	Length (m)	Width (m)	Height (m)	Area (m²)
Primary Crushing	Stick Built	14.5	12.8	20	186
Stockpile Cover	Geodesic Dome	130 (diameter)		53	13,273
Primary Grinding*	Pre-Engineered	104	96	32	9,984
Thickener Tanks and Leach Tanks*	Pre-Engineered	120	28	24	3,360
Tailings Leach and Filtration*	Pre-Engineered	88	40	24	3,520
Flotation and Reagents*	Pre-Engineered	64	40	24	2,560
Reagent Storage*	Pre-Engineered	24	32	15	768

* These are areas within the single concentrator building.

18.5 Site Infrastructure

The support buildings are listed in Table 18-3 below.

Table 18-3: General Facilities

Building Name	Construction Type	Length (m)	Width (m)	Height (m)	Area (m²)
E-houses (6)	Modular	4	15	3.5	60
Control Room (Substation E-Room)	Modular	27.8	6.3	3.5	175
Administration	Modular	27	21	3.5	567
Plant Warehouse	Pre-Engineered	40	32	15	1,280
Plant Maintenance	Pre-Engineered	40	40	8	1,600
Reagent Storage	Pre-Engineered	24	32	15	768
Plant Office/Laboratory	Modular	40	18.5	7	740
Gatehouse	Modular	6	4	3.5	24
Core Shack	Modular	26	26	3.5	650

Note: An allowance was made for the core shack facility. The footprint is based off similar-sized modular buildings and their allowed cost.

18.5.1 Accommodations

The Project requires a 1,569-bed accommodation facility to support construction. The facility includes an initial 200-person early works camp followed by an additional 1,369 beds added sequentially in advance of the peak of the construction period. The permanent operations camp will utilize 325 of the 1,569 beds through Phase 1 operation, with the remaining beds retained to support sustaining capital works, the Phase 2 expansion, and the increased needs for Phase 2 operating personnel.

To facilitate the construction of the expansion plant in year nine, 749 of the retained beds will be utilized for the construction crew. After completion of the expansion construction, the permanent camp will require 450 beds, thus 125 additional beds will be retrofitted for operations use through to mine closure.

The accommodation facilities will have individual rooms, washrooms and associated required facilities such as potable water and sewage treatment plants, kitchen and dining rooms, recreational areas, first aid facilities, camp offices and storage.

18.5.2 Sewage and Waste Management

Sanitary sewage will be collected from various areas of the site where it will be syphoned by vacuum truck and transported to a sewage treatment facility located at the plant and treated water will be pumped for reuse in process or for road dust suppression. The sludge will be transported to an off-site industrial treatment facility.

Waste bins will be supplied on-site at various work areas and personnel will be responsible for sorting their waste and hazardous materials into the appropriate bins for disposal. An incinerator will be used on-site for combustible waste disposal as there is no on site landfill. Any other material, including hazardous waste, will be transported off-site to regulated third-party disposal or recycling facilities.

18.6 Stockpiles

There will be two temporary stockpile facilities within the project footprint: the operational ore stockpile and the overburden stockpile.

The operational ore stockpile design will hold up to 30 Mt of rock with berm widths up to 100 m to allow access to the adjacent overburden stockpile. The operational ore stockpile will serve to stockpile ore generated during pre-stripping from the initial operating years. Stockpiled ore will be reclaimed to supplement open pit production maintaining the feed to the concentrator such that it is fully utilized. Any remaining material will be fed through the concentrator at the end of mine life, meaning the operational ore stockpile will be completely removed upon project completion.

The overburden stockpile will hold up to 50 Mt of overburden material. The majority of the overburden mined in pre-stripping and the early operating years will be used in tailings facility construction. The dump face angle is 37° (angle of repose) with 30 m safety benches placed at 25 m vertical intervals. This results in an overall slope angle of approximately 20°. The overburden material will be used for tailings facility reclamation purposes progressively during the mine life, hence the overburden stockpile will be completely removed upon project completion.

18.7 Tailings Facility

The proposed tailings facility is the key feature for the integrated waste and water management facilities, and will be progressively developed and operated over the LOM. The PFS mine plan includes four waste materials to be managed within, or for use in construction of, the integrated tailings facility:

- Tailings: tailings will be produced through processing of ore mined from the open pit and will include both non-magnetic tailings (consisting primarily of serpentine materials) and residue from the flotation tailings leach circuit (consisting primarily of magnetite). The combined process plant tailings will be discharged at a solids content of 25% w/w. The tailings facility will permanently store tailings produced from Years 1 to 27, while the final two years of tailings production will be stored within the open pit.

- Overburden: overburden from the open pit area will be removed for access to the ore body as mining progresses and the material will be used selectively for construction of the tailings facility and other site facilities or stockpiled for future use in tailings facility reclamation.
- Class A waste rock: Class A waste rock comprises below cut-off grade Trembleur ultramafite and Sowchea succession (dike) materials. Preliminary geochemical testwork indicates these materials are suitable for unrestricted use as a construction material owing to their very low potential for metal leaching or acid rock drainage (ML/ARD). This is detailed further in Section 20.1.2. Class A material will be used to construct downstream dam shells and buttresses.
- Class B waste rock: Class B waste rock comprises the Sitlika medasediments. Preliminary geochemical testwork indicates these materials may present some low potential for metal leaching, which is detailed further in Section 20.1.2. For the PFS, Class B materials will be managed within the tailings facility in a manner that will limit the potential for metal leaching conditions. Class B waste rock produced from Year 23 to Year 26 will be stored within the open pit.

The tailings facility dams will be constructed to El. 1,120 masl, with a corresponding maximum dam height of approximately 190 m (measured from the crest of the dam to downstream toe) and an overall slope of 3H:1V. The tailings facility will store a total of approximately 1 Bm^3 of tailings, overburden, and Class B waste rock produced through open pit mining and mineral processing.

18.7.1 Site Selection

A preliminary overall site layout has been developed to support the current PFS for the Project and is shown in Figure 18-1. The PFS tailings facility concept evolved from mine development trade-off studies completed between 2021 and 2022 and is currently proposed to be located outside of the Sidney Creek watershed and within the upper-most reaches of the Baptiste Creek catchment. The location of the PFS tailings facility was selected based on currently available technical and environmental criteria for the purpose of preparing the PFS. Early engagement with local communities and Indigenous rightsholders is ongoing and will continue to inform tailings facility siting assessments for future engineering and eventual environmental assessment and permitting reviews.

18.7.2 Facility Design

The tailings facility will incorporate three cross-valley dams situated near the open pit, within the upper reaches of the Baptiste Creek catchment, as shown on Figure 18-1.

The tailings facility will permanently store tailings and a portion of the overburden and Class B waste rock produced throughout the mine life:

- Tailings produced from mineral processing will be stored within the tailings facility from Year 1 to Year 27. The final two years of tailings production will be stored in the open pit.
- It is conservatively assumed that 25% of the overburden produced from Year -2 to Year 10 will be stored within the tailings facility. The material will be used to develop a causeway at a similar elevation to the dam crest, to be used for access to the reclaim pump barge. The remaining 75% of overburden material produced during this

period will be used for dam construction or tailings facility reclamation, as described in the following sections of this report.

- Class B waste rock will be stored in the tailings facility from Year 6 to Year 22. The material will be used to raise the proposed reclaim barge causeway across the facility. It is anticipated that some years of mining will generate more Class B waste rock than necessary for causeway development; during this time a waste rock dump adjacent to the causeway and overlying tailings will be developed. Class B waste rock produced from Year 23 to Year 26 will be stored within an in-pit waste rock dump. Class B waste rock produced from Year 1 to Year 5 will be used in dam construction.

The annual tonnages of material to be stored in the tailings facility over the 29-year mine life are shown in Table 18-4.

The North Dam will extend through Upper Baptiste Lake and surrounding area during initial construction. The depth to competent foundation material (yet to be defined but expected to be dense glacial till or glaciolacustrine soils) is not presently known; however, it is assumed for the present study that an average 5 m depth of soils will need to be removed from the dam footprint and will be stockpiled for reclamation. The West Dam, similarly, will cross several small water bodies and is expected to require excavation to reach competent foundation material. The depth of excavation for the West Dam during initial construction is assumed to be an average depth of 3 m within the dam footprint. The assumed depth of excavation beneath the dams is considered reasonable and appropriate; however, the assumption has not been verified through site investigation. The bulk of the foundation excavation work for the initial dam construction will be completed during Year -3 of pre-production.

The North Dam and West Dam will be initially constructed to El. 965 m during the pre-production phase of project development (Year -2 and Year -1) to impound water for mill start-up and provide containment for the first year of tailings production along with sufficient freeboard to store the inflow design flood (IDF). There is sufficient material available from pre-production stripping of the open pit for construction of the starter dams; however, most of the available material is overburden. It is conservatively assumed for the present study that 75% of the overburden material is suitable for dam construction, with the remaining 25% either containing too much organic content and/or having a moisture content too high to facilitate adequate compaction (not presently specified but expected to be equal to or greater than 95% standard proctor maximum dry density).

The dams will include a dedicated core zone, consisting of compacted low permeability glacial soils, to limit seepage and facilitate water retention during pre-production. An erosion protection layer of waste rock will be included along the upstream face of the dams, to protect the core zone from erosion during tailings deposition. The overburden used to construct the core zone will be sourced from ongoing stripping of the open pit as mining progresses.

Filter zones will be constructed downstream of the core zone to prevent internal erosion and piping, should any seepage flows develop through the core zone. These filter zones will consist of processed (crushed and screened) waste rock sourced from open pit mining. The filter zones will also act as a ‘chimney’ drain to transmit seepage flows to a network of foundation drains. The foundation drains will include primary rock drains constructed at the maximum sections (tallest dam height located at the lowest point within the topography) using coarse rock material sourced from open pit mining and will allow flows to be transmitted to locations downstream of the dam for collection, as well as secondary rock drains constructed to promote flow from localized low points within the dam foundations to the primary outlet drains.

Table 18-4: Tailings Facility Storage Requirements (KP, 2023)

Mine Year	Tailings			Overburden			Class B Waste Rock			Total
	Annual Total	Cumulative Total		Annual Total	Cumulative Total		Annual Total	Cumulative Total		
	(Mt)	(Mt)	(Mm³)	(Mt)	(Mt)	(Mm³)	(Mt)	(Mt)	(Mm³)	(Mm³)
-2	0	0	0	2	2	1	0	0	0	1
-1	0	0	0	3	5	3	0	0	0	3
1	30	30	23	3	8	4	0	0	0	27
2	39	69	53	2	10	5	0	0	0	58
3	39	108	83	1	11	5	0	0	0	89
4	39	148	114	1	12	6	0	0	0	120
5	39	187	144	0	12	6	0	0	0	150
6	39	227	162	0	12	6	2	2	1	169
7	39	266	190	1	14	7	0	2	1	198
8	39	306	218	1	15	7	4	7	3	229
9	39	345	246	0	15	7	8	15	7	260
10	53	398	284	0	15	8	9	24	11	302
11	59	457	305	0	15	8	20	44	20	332
12	59	516	344	0	15	8	21	65	30	381
13	59	575	383	0	15	8	13	78	36	427
14	59	634	423	0	15	8	7	85	39	469
15	59	693	462	0	15	8	1	86	39	509
16	59	752	502	0	15	8	0	86	39	548
17	59	812	541	0	15	8	0	86	39	588
18	59	871	580	0	15	8	4	90	41	629
19	59	930	620	0	15	8	13	103	47	674
20	59	989	659	0	15	8	14	117	53	720
21	59	1,048	699	0	15	8	11	128	58	765
22	59	1,107	738	0	15	8	9	137	62	808
23	59	1,166	778	0	15	8	0	137	62	848
24	59	1,225	817	0	15	8	0	137	62	887
25	59	1,285	856	0	15	8	0	137	62	926
26	59	1,344	896	0	15	8	0	137	62	966
27	59	1,403	935	0	15	8	0	137	62	1,005
28	0	1,403	935	0	15	8	0	137	62	1,005
29	0	1,403	935	0	15	8	0	137	62	1,005

Notes:
1. Tailings average dry density Year 1 to 5: 1.3 t/ ^{3}.
2. Tailings average dry density Year 6 to 10: 1.4 t/m^3.
3. Tailings average dry density Year 11 to 29: 1.5 t/m^3.
4. Rock and overburden by density 2.0 t/m^3.

The North Dam and West Dam will be constructed as downstream raises during the first years of mine operations (7 years of downstream raises are presently considered). The downstream raises constructed during this period will allow development of the facility without the need for support of the upstream zone on top of a tailings beach (as would be the case with a centreline raise). Dam construction will transition to centreline raises when drained tailings beaches have developed to support construction of a waste rock bench upstream of the core zone, serving the same purposes as the waste rock zone upstream of the starter core zone. Centreline raises of the dam will facilitate construction of a flatter overall downstream slope for the dam shell.

The footprint of the tailings facility is constrained to the west by Sidney Creek, to the north by the open pit excavation and to the east by Lower Baptiste Lake. The location of the starter dams considers a 200 m offset from each of these features and space has also been included to construct a buttressed downstream dam shell with an overall slope of 3H:1V. The minimum downstream slope of the dam will be 2H:1V, with the buttress constructed along the toe to create the overall flatter slope. The downstream dam shell will be constructed using Class A waste rock from open pit mining.

The timing of waste rock production from the open pit is ahead of the requirement for dam construction through the second half of the mine life, meaning that a full width buttress extending the ultimate footprint of the dam is expected to be constructed by Year 15 of operations. This early buttress construction will enhance the stability of the dams during operations and will facilitate monitoring of instrumentation to be installed within the foundation to confirm predictions of foundation behaviour during loading, and prior to finalizing design details for the ultimate dam configuration.

Detailed site investigation of the dam foundations and laboratory strength testing of materials representative of the foundations has not been completed and will be required to confirm the proposed staged development of the tailings facility dams for verification of stability and to meet regulatory requirements. Glacial deposits as found in the Project area have the potential to be highly variable and it is possible that weak glacial sediments may be present in some areas.

18.7.3 Tailings Distribution

Tailings will be conveyed by gravity from the plant site through two DR11 high-density polyethylene (HDPE) pipelines (diameter ranging from 36” to 48”) to the tailings facility for discharge to Year 6 of operations. The total length of pipeline installed during pre-production (Year -1) is approximately 7 km. A tailings pumping system will be installed and operated starting in Year 7 to support ongoing tailing deposition as the height of the tailings facility rises to and above the elevation of the plant site.

The tailings distribution pipelines will be added to facilitate discharge of tailings within the open pit in Year 28. The open pit dewatering system will also be upgraded in Year 28 to shift water reclaim primarily from the open pit rather than the tailings facility.

18.8 Waste Rock Storage Facilities

There are no permanent waste rock storage facilities presently required for the project; all mined waste rock and overburden is managed within the tailings facility, used to construct the tailings facility dams, or for reclamation material requirements.

18.9 Power and Electrical

18.9.1 Off-site Power and Distribution

The project will require 145 MW of power in the first nine years of operations (Phase 1), and up to an additional 74 MW in for the remaining LOM operations (Phase 2). A 155 km, 230 kV overhead transmission line, capable of transferring 250 MW, will be constructed to connect to the BC Hydro grid at the preferred Glenannan substation located 65 km west of Vanderhoof, BC. This point of interconnection has been assessed and is technically viable in a formal study completed by BC Hydro in January 2023. The study identified several upgrades to the substation and the BC Hydro transmission line network, which have been incorporated into the project cost estimate. Alternative transmission line routes have been evaluated, including a shorter eastern route that would connect to the BC Hydro grid at the Kennedy substation, also assessed by BC Hydro via their study and determined to be technically viable.

The transmission line route will avoid provincial parks, old growth forest areas, recreational sites, and known archaeologically sensitive areas. Wood pole H-Frame and three-pole dead-end structure types will be selected based on the terrain, design life of 30 years, and total cost to construct. The final corridor and route will be determined following discussion and collaboration with indigenous communities and property owners.

18.9.2 On-site Power and Distribution

Total site power consumption during Phase 1 is 1,015 GWh/a and will increase to 1,443 GWh/a in Phase 2. Electrical power will be received on-site at a 230 kV substation and distributed to two major electrical rooms including one for Phase 1 and a second added for Phase 2. Power will be stepped down to 34.5 kV, 4.16 kV, 600 V and 120/208 V distribution to accommodate medium and low voltage switchgears. The substation will be constructed in close proximity of the process area and utilize a similar two-phased approach matching the initial and expansion phase requirements. The electrical buildings and control rooms will be of a modular design, prefabricated, made of non-combustible material, suitable for installation on concrete piers, elevated from the ground.

18.10 Fuel

The on-site fuel storage will have a 10-day capacity including the mine and plant storage facilities. The tank farms will be provided for fuel storage with an appropriate fueling station. The fuel storage reservoirs will be contained within a bermed area and designed to meet applicable regulations. The mine fuel storage facility will have four tanks with a total capacity of 400,000 L of diesel and will be located along the haul road to the mine maintenance facilities. The plant fuel storage facility will have a single 50,000 L diesel tank and will be located along the site access road adjacent to the plant facilities.

18.11 Water Supply and Management Systems

18.11.1 Water Management Systems

The conceptual site water management plan for the Project includes collection of water in the tailings facility supernatant pond along with collection of runoff water downstream of all other project infrastructure/disturbance and then transfer to a process water pond at the plant site, where it will be reused in mineral processing.

Water management ponds will be constructed downstream of the tailings facility to collect surface runoff during and following construction of the North Dam and West Dam, along with seepage collected in the foundation drainage network. A single pond has been included for the North Dam (N1) and two ponds will be constructed for the West Dam (W1 and W2). The ponds will all be sized to store runoff from a 1- in-200-year storm event. A pumping system will be installed within each pond to match any seepage inflows and maintain a low water level, with fluctuation in a 4 m operating range only from seasonal runoff and storm events. An allowance is included for 1 m of dead storage (for sediment collection prior to periodic cleanout) and 1 m of freeboard above the maximum operating level.

The location of the tailings facility and downstream water management ponds at the upper end of the Baptiste Creek watershed limits the requirement for diversion. Water collection ditching will be constructed around the perimeter of the operational ore and overburden stockpile area to capture runoff upstream of Sidney Creek.

Water stored within the tailings facility pond will be reclaimed for use in mineral processing utilizing multiple barge mounted vertical turbine pumps. Two barges each holding four pumps will be installed during the initial 108 kt/d phase of operations, with a third barge installed as the throughput increases to 162 kt/d, to manage the resulting increased water available for reclaim.

An annual operational water balance has been developed for the tailings facility to assess the various flows to and from the facility, to evaluate the variation in supernatant pond volume during operations and to estimate makeup water requirements. The water balance indicates the site will be in a deficit condition throughout the operating life of the tailings facility. Fresh water pumped from Trembleur Lake will be used to maintain sufficient water for processing demands.

18.11.2 Open Pit Dewatering System

The estimated open pit groundwater inflow is 100 L/s at the time maximum mined depth is reached in the open pit. The groundwater inflow allowance, in addition to seasonal runoff, will be verified with additional hydrogeological testing of the open pit rock mass at future design stages.

The open pit dewatering system will remove runoff from normal precipitation events, groundwater inflows and runoff from any design storm events. Water collected with the open pit dewatering system will be transferred to the process water pond at the plant site for use in mineral processing. The design of the open pit dewatering system includes a diesel generator and collection sump pump at the base of the pit and a series of transfer pump stations within the pit. The water will be pumped out of the pit via transfer pump stations to the process water pond for reuse.

18.11.3 Freshwater Supply System

A trade-off study was completed to evaluate options for the freshwater supply source location. A range of potential water sources for the freshwater supply system were identified via a pre-screening assessment. Alternatives were selected by examining an approximately 20 km radius from the project area and identifying major features which could potentially provide a freshwater source. Through a pre-screening assessment, two alternatives (Trembleur Lake and Middle River) were advanced for further comparison. An economic evaluation of the two conceptual systems resulted in negligible difference in the anticipated capital cost of the systems (no difference within the accuracy of the TOS estimate). An intake at Trembleur Lake, south of the project area, is selected as the basis for the PFS, based on currently available technical and environmental information. A pump system will be installed on the shore of Trembleur Lake for supply of fresh water for mill operations and management of any potential water deficit on site.

Fresh water will be pumped from Trembleur Lake to a fresh water pond and gravity fed to the fresh water tank. The design of the freshwater pump and pipeline system includes a 14 km, 20" diameter covered on-grade steel pipeline operating at a constant pumping capacity year-round. The pumphouse will be constructed on the shore of the lake and a corridor comprising a maintenance road, the pipeline and power line will connect the pumphouse to the process plant. FPX will continue early engagement and environmental baseline data collection, to provide a basis to confirm the preferred design of the freshwater supply system and intake location. The freshwater supply system should also be optimized with the overall site water balance, to minimize the size of the system as much as reasonably practicable.

18.12 Hazard Considerations

KP completed an assessment of the regional seismicity and a preliminary probabilistic seismic hazard evaluation for the Project (KP, 2023). Seismic ground motion parameters determined by the probabilistic seismic hazard analysis indicate low hazard, with predicted peak ground acceleration (PGA) values of 0.02 g and 0.07 g for return periods of 475 and 10,000 years, respectively.

A desktop glacial geology assessment was completed for the Project (Clague, 2022). Inspection of Lidar and orthophoto imagery indicate the Baptiste Nickel Project area to be largely free of natural hazards; there are few very landslides, and all of them appear to be relict and likely thousands of years old. There is one significant stream within the project area, Sidney Creek and the study recommended that care must be exercised in any development on, or bordering, this floodplain because it might be inundated by rare extreme flows. The Austin FSR realignment road will serve a secondary purpose as an emergency flood levee for Sidney Creek, which runs north-south from west of the plant site area and past the West Dam of the tailings facility. The road will be armoured (where it runs alongside the creek) to divert extreme flows away from the West Dam area.

As described in Section 18.2.1, sections of the existing FSRs that would be used for project access will require upgrades to increase road strength and mitigate seasonal axle load restrictions to facilitate year-round hauling. Presently, this road system sees seasonal restrictions during spring melt. Road structure upgrades proposed will include the application of moisture wicking geosynthetics and high fines surfacing aggregate along 68 km of existing FSRs. The application of geosynthetic and high fines surfacing aggregate would also be applied along the proposed Middle River bypass segment and Austin FSR realignment.

18.13 Concentrate Bagging, Outloading, and Logistics

The awaruite concentrate will be processed into briquettes in the roller press and bagged into 90 cm x 90 cm x 127 cm (1 m^3) UN-rated bulk bags which will be put into trucking containers. The filtered MHP concentrate will be similarly loaded into 1 m^3 bulk bags but will require lined bags.

The briquettes will be packaged 2,000 kg per bag and then loaded in a standard 20 ft container with a typical maximum capacity of 24,947 kg. While larger bags are available, these smaller bags allow the containers capacity to be optimized and avoid blocking and bracing, which can be time-consuming and expensive.

Truck capacity is assumed to have a 45,720 kg payload. Each empty container is assumed to weigh 2,000 kg, therefore fitting 10 bags total per 20 ft container; each truck payload is approximately 40,000 kg of product.

An 8-axle Super B-Train with a tridem trailer chassis equipped for containers or flat deck will transport the briquettes to CN's export container hub in Prince George. Either truck configuration will be capable of taking two 20 ft containers per trip. In addition, a lighter chassis and more trailer axles may allow for additional payload capacity.

The container will then be transloaded onto an intermodal flatcar (well car) at the Prince George Distribution Center (PGDC). Two 20 ft containers typically fit on a single well car. The PGDC is serviced 7 days per week and has a track load limit of 129,727 kg (286,000 lb).

Container transport from Prince George to an assumed location of South Korea is handled through third-party shipping companies. Ocean container ships depart from Prince Rupert to South Korea weekly. In addition, the concentrate is securely stored in the containers for the duration of the export to South Korea, and no other special storage or transportation requirements are required.

19 MARKET STUDIES AND CONTRACTS

19.1 Introduction

The PFS Base Case outlines the production of two saleable products from the Baptiste Nickel Project:

- Awaruite concentrate grading 60% nickel for sale as a feedstock to stainless steel production.
- MHP grading 45% nickel for sale as a feedstock to nickel sulphate production for the electric vehicle battery supply chain.

Over the life of the Project, it is anticipated that the awaruite concentrate will account for approximately 93% of the contained nickel output, with the remaining 7% of nickel output attributable to the MHP product. This section describes how the selling price for the awaruite concentrate and MHP products was derived as the basis for the Baptiste Nickel Project economic analysis presented in Section 22.

19.1.1 Baptiste Awaruite Concentrate Product Overview

Metallurgical testwork, as described in Section 13, has shown that the Baptiste Nickel Project can produce a clean, high-grade, awaruite (Ni_3Fe) concentrate through a mineral processing flowsheet using conventional unit operations. The principal mineral recovered in the concentrate is awaruite, which is a naturally occurring alloy of nickel and iron. The concentrate will be dewatered, agglomerated into a briquette form, and bagged as described in Section 17 of this Report.

For the PFS Base Case, the intended application for the awaruite concentrate briquette is as a nickel additive for stainless steel production, similar to standard FeNi products.

Standard FeNi is a nickel-iron product that is almost exclusively used as a raw material in stainless steelmaking. Unlike pure nickel metal products (e.g., cathode), which are graded as Class I by the London Metal Exchange (LME), standard FeNi is a Class II nickel product which, by definition, means that it contains less than 99.8% Ni. FeNi products are not LME deliverable; consequently, FeNi has no terminal market and is typically sold directly from producers to stainless steel producers.

The price to be obtained from the sale of the awaruite concentrate briquettes to stainless steel producers will generally be a function of two variables: 1) the LME nickel price; and 2) a discount or premium to the LME nickel price, based on the market positioning of the Baptiste awaruite concentrate briquettes in relation to competing sources of nickel feedstock to stainless producers, being primarily stainless steel scrap, nickel pig iron (NPI), standard FeNi and Class 1 nickel.

19.1.2 Baptiste MHP Product Overview

Metallurgical testwork, as described in Section 13, has shown that the Baptiste Nickel Project can also produce a MHP product through atmospheric leaching of flotation tailings, followed by simple purification and precipitation unit operations. The tailings leach circuit will target very fine grained awaruite particles that are either unliberated following regrind or are too fine to be collected in flotation, resultantly improving overall nickel recovery. The MHP will be dried and bagged, as described in Section 17 of this report.

For the PFS Base Case, it is assumed that the Baptiste MHP product will be sold as a feedstock to producers of battery-grade nickel sulphate.

The nickel grade of the Baptiste MHP is expected to exceed 45% nickel. Baptiste MHP is expected to be higher grade than MHP produced from nickel laterite feedstocks via the high-pressure acid leach (HPAL) process route, which typically range 33-40% nickel. This is primarily due to the virtual absence of manganese minerals in the Baptiste Deposit which are a material contributor to impurity content in typical MHP derived from laterite feedstocks, thereby diluting nickel grades.

19.1.3 Other Potential Products

The PFS Base Case economic analysis assumes that 100% of the awaruite concentrate briquette product will be sold directly to stainless steel producers over the entire life of the project, and no byproduct credits are realized. The PFS Base Case economic analysis also assumes that 100% of the Baptiste MHP product will be sold directly to the battery material supply chain, and no byproduct credits are realized.

In Section 24 of this report, a refinery option is presented whereby a large portion of Baptiste's total nickel production would be directed to the production of battery-grade nickel sulphate for the electric vehicle battery supply chain. A discussion on the marketability of Baptiste's battery-grade nickel sulphate in the refinery option is presented in Section 24.

In addition to the PFS Base Case's primary awaruite concentrate and MHP products, the project has the potential to upgrade flotation tailings to produce an iron-rich byproduct that can potentially be sold as an iron ore concentrate on a standalone basis. For this PFS, the sale of this iron ore byproduct is not considered in the economic analysis, however, a discussion regarding this opportunity is presented in Section 25.

19.2 Characteristics of the Awaruite Product

The awaruite concentrate produced by the Baptiste Project is in the form of a fine powder/flake which will be agglomerated into briquettes. The chemistry of the Baptiste awaruite concentrate briquettes, based on PFS metallurgical testwork, is presented in Table 19-1.

Table 19-1: Production Specification for the Awaruite Concentrate Briquettes

Element	Unit	Value
Nickel, Total (Ni)	%	60
Nickel, Metallic (Ni)	%	~60
Iron, Total (Fe)	%	24.6
Iron, Metallic (Fe)	%	18.1
Sulphur (S)	%	0.98
Carbon (C)	%	0.10
Phosphorus (P)	%	<0.01
Silicon (Si)	%	2.2
Chromium (Cr)	%	0.06
Cobalt (Co)	%	1.08
Copper (Cu)	%	0.49
Magnesium (Mg)	%	1.7
Nickel:Sulphur Ratio (Ni:S)	Ratio	61
Nickel:Cobalt Ratio (Ni:Co)	Ratio	56
Nickel:Copper Ratio (Ni:Cu)	Ratio	122

In contrast to the awaruite concentrate briquettes, which are produced via a mineral processing route, standard FeNi and NPI are produced by pyrometallurgical routes and are highly metallized with carbon contents up to 5%. FeNi and NPI are typically produced from nickel laterite ores. Nickel content of such ores is typically in the range of 0.8% to 3.0% nickel. The predominant production route for ferronickel is the pyrometallurgical RKEF (rotary kiln – electric furnace) process. The RKEF process typically yields a standard FeNi product grading between 15% and 30% nickel. Laterite ores can also be processed to produce NPI. This blast furnace route can produce an NPI grading between 5% and 10% nickel. The submerged arc furnace (SAF) route can produce high-grade NPI grading 10% to 15% nickel. Table 19-2 presents a comparison of the nickel content for the various nickel products discussed.

Table 19-2: Nickel Content Comparison of Various Nickel Products

Element	Baptiste Awaruite Concentrate	Class 1 Ni	Standard FeNi	High Grade NPI	Low Grade NPI
% Ni	60%	≥99.8%	20% - 35%	10% - 15%	5% - 10%
% Fe (metallic)	18%	NIL	balance	balance	balance
% Other	22%*	NIL	1% - 4%	5% - 10%	7% - 13%

*Consists mainly of magnetite Fe_3O_4 and other oxide minerals.

As shown in Table 19-2, the awaruite concentrate briquette has some unique characteristics compared to other products, most notably its high nickel content. The awaruite concentrate briquette has relatively high purity; after Ni_3Fe, the next most abundant component in the briquette is iron oxide in the form of magnetite. Review of the Baptiste awaruite concentrate specification with a leading stainless steel producer confirms that it would be directly useable in the stainless steel electric arc furnace process without having to go through an intermediary smelting or refining process. The awaruite concentrate briquette could thus be used to substitute or complement other forms of nickel, including standard FeNi, in stainless steel production.

19.3 Global Nickel Supply and Demand

Finished nickel products are, today, primarily used as an alloying agent in the production of stainless steel, superalloys and a variety of nickel-copper alloys, plating material on steel and plastics, as well as in batteries for electric vehicles and other applications.

Global nickel consumption in 2022 reached 3.0 Mt nickel, up 4% from 2.9 Mt nickel in 2021 following a significant increase in demand the year before. CRU forecasts consumption growth to continue to grow robustly at a rate of 12% in 2023 reaching approximately 3.4 Mt nickel. This strong growth is expected to continue, driven by substantial demand from the electric vehicle sector and stainless steel production. Stainless steel production is expected to remain the most important end-use sector for finished nickel products, accounting for 63% of total primary nickel consumption in 2022.

Stainless steel producers have three sources of nickel for their furnaces: recycled stainless steel scrap, nickel-iron alloy (standard FeNi and NPI) and Class 1 nickel. The availability of these three primary nickel feedstocks is highly regional and subject to local and global supply-demand dynamics at a given time. As such, the use and pricing of scrap, standard FeNi, NPI and Class 1 nickel are interrelated and interdependent.

19.4 LME Nickel Price for PFS Economic Analysis

FPX provided long term projected nickel price data published by several reputable analysts as of August 2023. A long-term nickel price assumption of $19,290/t (US$8.75/lb) is assumed in this PFS Base Case, which is consistent with the average long-term nickel price forecast published by six base metals analysts, as presented in Table 19-3.

Table 19-3: Projected Analyst LME Benchmark Price

	Projected Price	
	US$ / lb Ni	**US$ / t**
Analyst 1	8.00	17,637
Analyst 2	8.00	17,637
Analyst 3	9.00	19,842
Analyst 4	9.00	19,842
Analyst 5	9.00	19,842
Analyst 6	9.50	20,944
Average	**8.75**	**19,290**

19.5 Baptiste Product Payability Analysis

The following sections discuss the payability analysis for Baptiste's awaruite concentrate and MHP products. A discussion of payability analysis for the Refinery Option's nickel sulphate product is presented in Section 24.

19.5.1 Baptiste Awaruite Concentrate Product

CRU, a leading provider of analysis and consulting in the mining, metals and fertilizer markets, prepared a market analysis report that looked at the FeNi market and considered the applicability of the Baptiste awaruite concentrate briquette to stainless steel production and the comparability of the Baptiste awaruite concentrate briquette to standard FeNi. The analysis utilized CRU in-house nickel experts, CRU CO2e sustainability data, FPX metallurgy and product specialists, and discussions with a global stainless steel producer and a global nickel product trading firm. Expertise in mining, steelmaking, supply chain, purchasing and marketing was all utilized to support the CRU analysis.

In general, the high total percentage of nickel and metallic iron in the Baptiste awaruite concentrate should provide an overall advantage for the product, allowing for less of it to be used per every given unit of nickel. This will reduce issues with unwanted secondary elements as well as minimizing transportation costs and Scope 3 carbon dioxide emissions.

The high nickel content of the Baptiste awaruite concentrate will reduce the amount of secondary elements charged into the melt. Simply put, the Baptiste awaruite concentrate's high nickel content requires only one-half to one-third the amount of traditional FeNi products, which in turn means that the burden of secondary elements will be only one-half to one-third that of traditional FeNi materials. Further, the high nickel content of the Baptiste product will allow for the minimization or elimination of the more expensive Class 1 nickel typically used for trim in the stainless steel production process.

The Baptiste awaruite concentrate will be sold in the form of briquettes. While most (~70-80%) FeNi is sold in granulated form, briquettes are also commonly used, and thus it is unlikely that this physical form will have any effect on product payability.

Regarding secondary elements in the Baptiste awaruite concentrate (summarized in Table 19-1), silicon (2.2%), sulphur (0.98%), and copper (0.49%) represent the elements with the highest potential concern from the standpoint of downstream application in stainless steel. Based on the CRU analysis, the issues with silicon and sulphur are considered minor, as these elements are generally present in other competing feedstocks (such as standard FeNi and NPI) and are routinely removed from the melt during the steelmaking process. Copper, however, cannot be removed and will remain in the melt. However, as copper does not cause hot-shortness in stainless steel, it is not considered a significant issue, and given the similarly elevated levels of copper in competing feedstocks like stainless steel scrap, it is not anticipated that the copper content in the Baptiste awaruite concentrate will have a negative impact on payability.

An additional positive for the Baptiste awaruite concentrate is its very low-carbon footprint. FPX's internal evaluation for the Baptiste Base Case Scope 1 and 2 carbon intensity indicates approximately 2.4 t CO2e/t of nickel, which would place the product in the lowest decile of Skarn Associates' database of global nickel producers. For steelmakers looking to reduce their carbon footprint, the Baptiste awaruite concentrate briquette would be an attractive product versus more carbon-intensive feedstocks. For the purposes of this Report, no additional value or payability premium has been attributed to reflect the low-carbon nature of the Baptiste awaruite concentrate.

Overall, CRU determined the Baptiste awaruite concentrate should be able to substitute for standard FeNi in stainless steel production, and therefore attract similar payability to standard FeNi. Table 19-4 presents historical premium/discount data for standard FeNi.

Table 19-4: Historical Premium / Discount for Standard FeNi

Year	Average LME Price (US$ /t)	Average Realized Price (US$ /t)	Premium / Discount (%)
2017	10,406	10,494	+1
2018	13,118	12,963	-1
2019	13,933	13,757	-1
2020	13,779	12,412	-10
2021	18,497	17,042	-8
2022	25,596	22,619	-12
Mean			**-5**
Median			**-4**

Source: Anglo American annual reports.

It should be noted that the cost of transport from the mine site to the destination port is included in the operating costs of the product; therefore, no adjustment to payability has been made to account for product transportation. Transportation costs are discussed in Section 21 of this report.

19.5.2 Baptiste MHP Product Payability Analysis

Based on publicly available market data, the payability for nickel content in MHP ranged from 70% to 90% of the LME nickel price over the 2020-22 period, with the low-end of that payability range coinciding with the period of extreme market volatility and elevated LME nickel prices in the first half of 2022. For the purposes of the economic analysis in Section 22 of this report, payability of 87% of the LME nickel price has been assumed for the Baptiste MHP product.

19.6 Selling Prices Used for PFS Economic Analysis

As mentioned previously, the LME price and payability factor (expressed as a discount or a premium) for FeNi products is dependent on many factors driven mainly by the stainless steel scrap market and the availability of standard FeNi and NPI products on a regional and global basis at any given time. Historically, nickel values have fluctuated both in terms of the LME price and in terms of the discount or premium paid for FeNi and NPI products. These fluctuations arise as a result of shifts in supply and demand and geographic variation in feedstock availability.

Based on the average projected analyst LME benchmark nickel price and the payability analysis presented earlier in this chapter, Table 19-5 provides the basis for the selling prices assumed in the Economic Analysis presented in Section 22 of this Report.

Table 19-5: Selling Price for Nickel Products in PFS Economic Analysis

	Awaruite Concentrate Product $/t Contained Ni	MHP Product $/t Contained Ni
LME price based on analyst average	19,290	19,290
Payability (% of LME price)	95%	87%
Selling price used in PFS Economic Analysis	**18,326**	**16,782**

19.7 Contracts

In May 2023, FPX granted a contractual right of first offer which provides the major global stainless steel producer Outokumpu Oyj the right, exercisable under certain conditions, to negotiate, at market terms, one or more offtake agreements with FPX for the purchase up to an aggregate of 60 kt Ni (7.5 kt Ni per annum over a period of eight years) in the form of awaruite concentrate briquettes from the Baptiste Nickel Project.

Other than as noted above, FPX does not have any contracts, agreements or any commercial engagements with regards to the sale of the Baptiste nickel products or to the development of the Project.

19.8 Qualified Person Comments

The QP has reviewed these studies and analyses, and the results support the assumptions used in the technical report.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Baseline Studies

The Baptiste Nickel Project has benefited from environmental baseline studies completed during two distinct periods: from 2011- 2014, and 2021- present. The 2011-2014 studies were led by Cliffs Natural Resources, in parallel with project development activities at the time associated with the 2013 PEA. In 2014, Cliffs announced the intention to divest from the project, and baseline data collection ceased accordingly.

In 2021, following issuance of the 2020 PEA, FPX began planning efforts to resume baseline study data collection. FPX has engaged a local (Fort St James based) Indigenous-owned cultural and environmental consulting firm, Shas-Ti Environmental LP, to lead the cultural and environmental baseline studies for the Baptiste Nickel Project. FPX also engaged ERM Consultants Canada since 2021 to undertake specific environmental baseline studies for meteorology, hydrology and surface water quality supported by local Indigenous field-assistants during fieldwork. Advancement of the project through the environmental assessment (EA) review process will need to ensure that all baseline studies have been conducted in a manner that addresses both local study area and regional study area definition for the project footprint, including off-site infrastructure.

20.1.1 2011 to 2014 Baseline Studies

Preliminary environmental baseline studies were conducted in 2011 and 2012 by KCB (KCB 2012). Individual studies were continued and expanded in 2012, led then by Allnorth Consultants Ltd (Allnorth 2013a). Key study scope components for this period are summarized in Table 20-1.

Table 20-1: Summary of 2011-2014 Environmental Baseline Studies

Study Area	Scope of the Study
Climate (Allnorth, 2013b, 2014b)	• A meteorological monitoring station was installed in 2012 and was equipped with sensors for wind direction and speed, precipitation (tipping bucket), temperature, relative humidity, snow depth and water equivalent (ultrasonic), and evaporation.
Hydrology and Surface Water Quality (Allnorth, 2013c, 2013d, 2014c)	• The Decar mineral claims are located in the Stuart-Takla watershed and encompass three sub-watersheds associated with Baptiste Creek, Sidney Creek and Van Decar Creek. Van Decar Creek and Baptiste Creek flow into Middle River approximately 18 km and 5 km, respectively, northwest of Trembleur Lake; Sidney Creek flows south from the western side of Mt. Sidney Williams, directly into Trembleur Lake. The outflow from Trembleur Lake flows southeast into Stuart Lake, which is the lowest catchment of the Stuart-Takla watershed. • Baptiste Creek is the main watercourse within the Baptiste Project area. Baptiste Creek receives water from Upper and Lower Baptiste Lakes, two small headwater lakes and their tributaries, Camp Creek and Nickel Creek, which drains Nickel Lake. • Monthly water quality monitoring was undertaken at 10 to 15 sites along the main watercourses and their related tributaries during 2011 to 2013. • Manual flow monitoring has been undertaken (irregularly) at the surface water monitoring stations. • Three hydrometric stations, consisting of a Solinst pressure transducer and stream stage gauge, were installed October 2011.

Study Area	Scope of the Study
Hydrology and Groundwater Quality (Allnorth, 2013e, 2014d)	• Eight hydrogeologic monitoring wells were installed in September 2012 and configured with vibrating wire piezometers or data loggers, as appropriate. • Lugeon packer tests and/or falling head tests were undertaken in the hydrogeologic monitoring wells to determine hydraulic conductivity within the proposed pit area. • Water quality and groundwater levels were collected from five non-vertical, open exploration drillholes in October 2011.
Acid Rock Drainage/Metal Leaching (Caracle Creek, 2012)	• 218 samples of representative overburden, ore, and waste rock lithologies were sampled and analyzed for a combination of static acid base accounting parameters, metals, and shake flask extraction.
Vegetation (Allnorth, 2013f, 2013g)	• Review of existing information from recognized government databases was completed. • White bark pine surveys, including geo-referencing and flagging, were undertaken over the areas impacted by the 2011/2012 drill area. • Vegetation surveys over the Project footprint were undertaken to develop a database of rare and invasive plant information, plant tissue, and Terrestrial Ecosystem Mapping ground truthing data.
Terrestrial Wildlife (Allnorth, 2014a)	• Review of existing information from recognized government databases was completed. • Wildlife habitat mapping has been undertaken. • Wildlife transects were seasonally completed, as appropriate, to determine the presence of mammal, bird, and amphibian species. • A winter caribou survey was undertaken and discussions with regulators regarding caribou monitoring and management commenced.
Aquatic Environment (Allnorth, 2013h, 2013i)	• Fish population data and stream inventory (fish habitat) assessments were undertaken in Nickel Creek, Camp Creek, Baptiste Creek, Sidney Creek, Van Decar Creek, Upper/Lower Baptiste Lake and Nickel Lake. A limited number of opportunistic fish tissue samples were collected and analyzed. • Spawning assessments were undertaken in Baptiste Creek and Van Decar Creek. • Sediment quality sampling was undertaken. • Periphyton, phytoplankton, zooplankton, and benthic invertebrate sampling were undertaken.
Archaeology (Allnorth, 2013j)	• An archaeology overview assessment and preliminary field reconnaissance survey was conducted for the 2011/2012 drillhole locations.

20.1.2 2021 to Present Baseline Studies

The current and in-progress baseline program was re-initiated during the summer of 2021, continued through a 2022 summer program, a 2022/2023 winter program, and currently a summer 2023 program is underway at the time of writing this report. The baseline studies are planned to continue for the remainder of 2023 and into 2024. These baseline studies are in support of future environmental/impact assessments and permitting for the Project, however the Project has yet to enter the EA process formally. At that time, it is anticipated that future direction will be obtained on the completion of all applicable baseline studies.

FPX has engaged a local, Fort St James based, Indigenous-owned cultural and environmental consulting firm, Shas-Ti Environmental LP, to lead the cultural and environmental baseline studies for the Baptiste Nickel Project. Shas-Ti has incorporated Traditional Ecological Knowledge to develop and implement the wildlife, fisheries, aquatics, and vegetation baseline programs, and the programs have been designed to maximize local capacity and training opportunities. Special emphasis of the program has been inclusion of local Indigenous field assistants in conducting the studies and incorporation of traditional knowledge elders have for the region, to support development of the scope of the studies. Commencing in 2021, and continuing through 2023, ERM re-initiated baseline studies of atmospherics, surface water quality and hydrology. The study components for this period are summarized in Table 20-2.

Table 20-2: Summary of 2021-2023 Environmental Baseline Studies

Study Area	Scope of the Study
Atmospherics (ERM, 2023a)	• Initiated in 2021 with re-instrumentation of the existing climate station, originally installed in 2012. The meteorological station is operating at the site recording continuous weather conditions to support engineering design studies, air quality dispersion modelling, hydrologic studies, and water balance modelling. • The station measures air temperature, total precipitation, snowpack depths, wind direction, solar radiation, barometric pressure, and the evaporation rate. The station provides continuous remote monitoring for quality assurance checks on the data collection instruments. • The meteorological station will remain in operation for the life of the project.
Aquatics (Shas-Ti, 2023a; ERM 2021).	• The aquatics program has been developed based on guidance through the Provincial government's Water and Air Baseline Monitoring Guidance Document for Mine Proponents and Operators (BCMoE 2016) as well as the federal government's Metal Mining Technical Guidance for Environmental Effects Monitoring (EEM) guidelines (EC 2012), and (3) other pertinent scientific grey literature. The development of this program has also considered findings/recommendations from previous fisheries-related surveys and assessments conducted in the Study area (Ecofor 2011, Allnorth 2013, ERM 2021). • Surface water quality, phytoplankton, zooplankton, benthic invertebrates, sediments, and macrophyte tissues were collected at lentic sites throughout the study area. Lake surveys were also completed to determine bathymetric characteristics, stratification, and maximum depths. • The lentic and lotic aquatics programs are planned to be repeated in 2023 in continuation of the data collection and reoccur annually afterwards. However, it is anticipated adjustments will occur to the frequency and/or location of sampling efforts (e.g., some may be deprioritized) as the mine layout is further developed and environmental assessment (EA) objectives are refined.
Archaeology (Ecofor, 2023)	• Reconnaissance studies have been conducted across the study area during multiple previous field seasons by various consultants for forestry, oil and gas, and the Baptiste Nickel Project, including an archeological overview assessment conducted in 2013. The 2022 program built upon these previous findings and documented additional areas of interest. There are 38 untested areas of potential (AOPs) identified within the project footprint with potential for subsurface archaeological materials. The remainder of the areas traversed during the PFR were assessed to have low archaeological potential. The assessment of the Project general arrangement footprint is incomplete. Further archaeological impact assessment is required for the entire project footprint. Details with respect to identified archaeological sites are confidential and only shared with the appropriate First Nation and the provincial government.
Fisheries (Shas-Ti, 2023b; ERM 2021)	• Fisheries studies focused on the primary drainages within the project area. Baptiste Creek (Khast'ani koh), Sidney Creek, Paula Creek (Tse'ahunin koh), and Nickel Creek. Lakes studied included Upper Baptiste (Ude'az), Lower Baptiste (Khast'anghunbun) and Nickel Lake. • Surveys included spring spawning, 1:20,000 habitat reconnaissance, fish communities in lakes and creeks, fish tissue, and fall spawning. • Spring spawning surveys identified Rainbow Trout and Longnose Sucker at Upper Baptiste Creek (Khast'ani koh) and Nickel Creek, while Lower Baptiste Creek (Khast'ani koh) produced Bull Trout, Peamouth Chub, and Mountain Whitefish as well. • During the habitat reconnaissance program, a total of 148 sites were surveyed within the study area. Final classification of streams will be assigned once analyses are complete including findings obtained from a combination of mapping exercises, additional sampling, and eDNA results.

Study Area	Scope of the Study
	• As part of the fish community program, a total of 39 sites were assessed. Typically, watercourses that meet the definition of a stream would then be assigned as an S-1 to S-4 (fish bearing) and S-5 and/or S-6 (non-fish bearing) watercourse. Given analyses are not complete at this time, watercourses were only characterized based on channel width and confirmed fish presence/absence. Three species were encountered during fish sampling efforts: (1) Rainbow Trout (Oncorhynchus mykiss), (2) Bull Trout (Salvelinus confluentus), and (3) Longnose Sucker (Catostomus catostomus). Rainbow Trout was the most common species, Bull Trout was only documented in Nickel, Paula (Tse 'ahunin koh) and Sidney creeks, while Longnose Sucker was only captured in Baptiste Creek (Khast'ani koh). A permanent barrier was also noted in lower Sidney Creek (~4 km) upstream from the confluence with Trembleur Lake (Dzinghubun) that hinders upstream fish migration. • Three lakes were included in the fish community lake program: Nickel, Upper Baptiste (Ude'az), and Lower Baptiste (Khast'anghunbun). As with the Fish Community-streams program Rainbow Trout, Bull Trout, and Longnose Sucker were encountered. Rainbow Trout was the most abundant species in the waterbodies sampled. Notable captures include a large mature Bull Trout (455 mm, and 989 g) in Lower Baptiste Lake (Khast'anghunbun), and the presence of numerous Longnose Sucker in Upper Baptiste Lake (Ude'az) relative to Lower Baptiste Lake. • Using fish presence data collected during the fish community field programs, tissue samples were collected from Rainbow Trout (most widely distributed sportfish) and Longnose Sucker (most abundant coarse species). A total of 87 fish from eight watercourses/waterbodies were submitted to ALS laboratories for metals analysis (Table 5). For large individuals (>150 mm FL), analyses were performed on muscle tissue and liver, while small individuals were analyzed as whole-body samples. • Fall spawning surveys were conducted using a combination of aerial (i.e., helicopter) and ground based (i.e., foot) methods occurring between late August and early October. The upper distribution extent of respective fall spawning species and any potential migration barriers were recorded on Nickel, Baptiste (Khast'ani koh), Sidney, and Paula (Tse 'ahunin koh) Creeks. Four Bull Trout were observed in the lower section of Nickel Creek near the confluence with Lower Baptiste Lake (Khast'anghunbun). Timing, fin coloration and presence of mature fish suggest that there is a high potential that spawning occurs in Nickel Creek. • Multiple, large, non-permanent obstacles (beaver dams) were observed in Lower Baptiste Creek (Khast'ani koh) approximately 1.1 km upstream from the confluence with Middle River (Yoonoo'-I Koh). These large obstructions are believed to impede upstream migration of Kokanee and Sockeye Salmon. A permanent barrier (waterfall/chute) was identified on Sidney Creek approximately 4 km upstream from the confluence of Trembleur Lake (Dzinghubun), though spawning salmonids were only observed immediately upstream (~300 meters) from the Trembleur Lake (Dzinghubun) confluence. Similarly, fall spawning on Paula Creek (Tse 'ahunin koh) appears to only occur within the first ~600 m upstream of the Trembleur Lake (Dzinghubun) confluence. Further investigations in 2023 and 2024 will be required to confirm information gathered to date and account for any annual variability in spawning runs.
Geochemistry (Lorax, 2023)	• Lorax Environmental Services Ltd. was retained by FPX in 2022 to initiate a geochemical characterization of ore, overburden, and waste rock as a basis to develop subsequent management plans for these materials. • The 2022 program prioritized the collection of 202 waste rock samples from 30 drill holes. This program was intended to fill spatial and lithological gaps that existed in the 2012 studies, with the assistance of the FPX Geology department that provided modelling results. Twelve different lithologies were sampled from drill core stored in Fort St James. • Acid base accounting results indicate that waste rock shows low to moderate total sulphur concentrations with a median of 0.035%. Samples from all geological units show neutralizing potentials greater than 2 and are considered non potentially acid generating. • Shake flask extraction results indicate that metal parameters with particularly elevated concentrations include As, Cu, and Se for one waste rock type (Stilika Assemblage).

Study Area	Scope of the Study
	• In a preliminary assessment completed to provide recommendations for the PFS tailings facility design and waste rock management, Lorax has identified two management classes for waste rock materials: Class A, designated for unrestricted use in the construction of the tailings facility embankment due to a very low potential for ML/ARD and Class B, comprised of the Sitlika Assemblage; it is recommended at this time that Class B material be managed within the tailings facility in conditions that will limit contact with oxygen, as much as is operationally feasible for the PFS.
Hydrology and Surface Water Quality (ERM, 2023b; ERM, 2023c)	• Surface water flow and water quality is currently being monitored monthly at the Baptiste site to establish baseline conditions. Surface water quality monitoring is conducted at eight stations with seven of the stations being co-located with hydrology monitoring stations. In situ readings were collected for water temperature, pH, conductivity, specific conductivity, dissolved oxygen (concentration and saturation), and turbidity. Laboratory analysis for water samples was conducted for physical tests (total suspended solids [TSS], total dissolved solids [TDS], alkalinity, hardness, etc.), total and dissolved nutrients, total organic and inorganic carbon, anions (chloride, fluoride, bromide, sulphate), and total and dissolved metals. Samples underwent a QA/QC process and were screened against applicable federal and provincial guidelines for the protection of aquatic life. • Annual discharge hydrographs are being generated for each of the hydrometric monitoring stations operated in 2022 and into 2023. For the operational period at each hydrometric station, water discharges were calculated at ten-minute intervals by applying the developed rating curve equation to the recorded stage data. The ten-minute discharge data were averaged over a 24-hour period to calculate mean daily discharge. Discharge data was used to generate hydrologic indices for each station.
Vegetation (Shas-Ti, 2023c)	• The vegetation field surveys were conducted to support the anticipated development of an environmental assessment application and included terrestrial ecosystem mapping (TEM), rare plants and communities, noxious weeds, and plant tissue chemical analysis. The vegetation study area aligns with the wildlife study area. • Whitebark Pine occurrences were recorded incidentally during traverses between TEM plot sites and/or work for other project components. The overall proportion of trees with visual evidence of blister rust was also noted. In general, Whitebark Pine is locally common at higher elevations, forming the dominant tree species in sub-alpine parkland areas and a minor component of the forest at lower elevations. No rare plants, with the exception of Whitebark Pine, were identified during the TEM survey plots or during traverses between survey plots. Fifty-eight plant specimens were collected to confirm species identification (species identification ongoing). • Seven species of noxious plants were identified in 19 mapped locations along existing roads and within regenerating cut blocks, representing large populations of invasive species, especially Oxeye Daisy and hawkweeds. • Further field data collection is required to support the preparation of a full existing conditions characterization and subsequent environmental assessment.
Wildlife (Shas-Ti, 2023d)	• The wildlife baseline data collection program commenced in 2021 and is focused on the Species at Risk, species valued by local First Nations, species known to be sensitive to landscape changes, and species of regional or management importance. Preliminary findings of the historic monitoring program with confirmation by the 2022 field season include the following observations. • High and low elevation habitats are utilized by caribou during summer and autumn. Current data suggests that there are a minimum of four caribou in the study area, as evidenced by four unique sets of tracks that were observed in parallel to each other on Tselkun (Mount Sidney Williams). Photographic evidence of a caribou cow and calf also suggests the presence of a breeding population in this area. • Initial study results suggest that moose appear to be broadly distributed at low elevations within the study area and are the most abundant ungulate in the area. Other ungulates in the area include elk, mule deer, and white-tailed deer. No observations of elk, mule deer, or white-tailed deer calves were recorded although adult females of all species were observed. These preliminary results suggest that the area provides low-quality post-calving habitat for these species.

Study Area	Scope of the Study
	• A minimum of six grizzly bears were captured on remote cameras, including three sows with cubs. Grizzly bears were broadly distributed in the study area, including within components of the proposed project footprint. • Preliminary results suggest that wolverine and fisher species are broadly distributed in the study area and within the components of the proposed project footprint. Fisher were located in low elevation habitats, primarily within stands of old and mature forest. Wolverine were located on Tselkun and in low elevation habitats. Canada lynx detections suggest broad distribution within the study area including within components of the proposed project footprint as well as olive-sided flycatcher. Evening grosbeak were recorded on 5 occasions but were not detected within the proposed project footprint. Western toads were not detected during targeted amphibian surveys; however, the species was seen incidentally on 34 occasions, primarily within the project footprint. • Subsequent programs will need to be conducted to capture seasonal variability in habitat selection and distribution. Results from winter study programs (2023) are pending, and summer 2023 programs will fill data gaps that exist to support pending environmental assessments.

20.1.3 Outstanding Areas of Study Required to Fulfill the Environmental Assessment Requirements

At such time as the Project commences the EA process, FPX will be informed through the development of the EA Application Information Requirements (AIR) of the required application information contents, including the range of baseline studies, modelling, effects assessment and management plans required to advance the EA Application. Based on prior recent experience of mining projects undergoing an EA in BC, FPX has identified the following key studies, which may not be a complete list, additional to those listed in Table 20-2 which will need to be undertaken:

20.1.3.1 Surface and Groundwater Modelling

FPX has enlisted the support of Knight Piésold for life of mine water balance modelling, Lorax for geochemical characterization, ERM for atmospherics and surface hydrology baseline programs, and Sasuchan Environmental LP (SELP) for groundwater characterization and modelling. These five aspects will be integrated into an overall watershed model. Establishment of groundwater monitoring wells upstream and downstream of planned project infrastructure is required to characterize hydrogeologic conditions and support development of an integrated life of mine water balance and receiving environment (watershed) model.

20.1.3.2 Socio-Economic Analysis

The environmental assessment process will need to be supported by a local socio-economic analysis. The EA AIR will specify the scope of the socio-economic baseline study and effects assessment and the timeline for initiation will proceed as early as 2024 as guided by the EA AIR development.

20.1.3.3 Traditional Knowledge and Land Use Study

The environmental assessment process will need to be supported by a local Traditional Knowledge analysis. Early engagement has been initiated by Shas-Ti and will continue throughout the 2023 and 2024 seasons. The Traditional Knowledge and Land Use Study scope of studies and analysis will be guided by development of the EA AIR collaboratively with Indigenous communities.

20.1.3.4 Human Health and Country Foods

Human health, ecological risk assessment and country foods baseline studies and analysis of the effects of the Project on these valued components will need to be undertaken in accordance with the approved AIR scope during the EA. FPX has engaged a specialist to review and recommend areas to focus on, or in addition to, the current vegetation survey baseline programs which will contribute to the full scope of these studies.

20.1.3.5 Baseline Studies of Linear Features

Once the footprint of the two major linear features (main access road and powerline) associated with the Project are finalized, each of the environmental disciplines listed above will be required to conduct baseline studies and identify potential impacts. A cultural assessment for each of the ancillary facilities will also have to be conducted.

20.1.3.6 Air Quality and Noise

Air quality and noise baseline studies will be initiated during the 2024 field season and inform the project environmental setting, analysis of the Project's effects on the environment, and mitigation through engineering design of important effects.

20.2 Environmental Assessment and Permitting

The environmental assessment and permitting framework for mining in Canada and British Columbia is well established and prescriptive with supporting guidance documents provided by government agencies. Currently, cultural and environmental baseline studies are ongoing to provide current conditions to inform the pending Provincial Environmental Assessment, Federal Impact Assessment, and permit applications. Individual permits to construct and operate the mine will only be approved for the Project subsequent to the issuance of a provincial Environmental Assessment Certificate and the issuance of a federal Decision Statement.

20.2.1 Environmental Assessment Process

20.2.1.1 Provincial Process

In BC, the *Environmental Assessment Act* (2018) defines whether a mine project is a reviewable project under the Act and Regulations if it exceeds certain thresholds. The Reviewable Projects Regulation (BC Reg. 67/2020), Part 3 – Mine Projects, Table 6 specifies that a new mine is a reviewable project if the annual production will exceed 75,000 tpy of mineral ore. This Project exceeds the threshold criteria and therefore is subject to review under the BC *Environmental Assessment Act* and Regulations. The Project must obtain an Environmental Assessment Certificate (EAC) prior to obtaining the required construction and operating permits. The BC *Environmental Assessment Act* review process uses a phased approach with imposed regulatory timelines. The EA is generally structured as follows:

- Identification and assessment of potential environmental, social, economic, cultural, and health impacts.
- Development of an acceptable scope and methodology for conducting the effects assessment of a selection of valued components (VCs).

- Characterization of residual effects potential for VCs after avoidance, mitigation measures, standard best management practices (BMPs), and monitoring programs are implemented.
- Prediction of the likelihood of significant residual effects occurring.
- Development of acceptable compensation measures to offset residual effects and maintain compliance with provincial and federal regulatory requirements as well as to effectively accommodate adversely affected Indigenous groups.
- Participation in Crown consultation proceedings to provide opportunities for Indigenous, federal, provincial, and local governments, stakeholders, special interest groups, and members of the public to learn about the project, identify potential issues, provide input to potential avoidance/mitigation measures, and accommodate any infringement of Indigenous title and rights.
- Incorporate economic, social, cultural, health, and environmental factors into proponent and government decision making processes.

Key phases of the EA process in BC include:

- Early engagement phase (minimum of 90 days) occurring prior to submitting the detailed Project Description (s. 13 and s. 15 of the Act) and designed to attain consensus among participating Indigenous groups. It includes alternative dispute resolution options and leads to a Summary of Engagement and the Detailed Project Description.
- Remaining phases required to obtain an EAC include:
  - EA readiness phase and decision (s. 16(2), s. 17 or s. 18; 60 days minimum, but timeline is variable);
  - Process planning phase (120 days);
  - Application development and review phase (minimum of 180 days) and submission of final application;
  - Effects assessment and recommendation phases (150 days maximum); and
  - Decision phase (30 days maximum).

The project will be bound by the conditions of the EAC. Post-certificate activities include mitigation effectiveness reports and may include audits, certificate amendments, extensions, and transfers.

20.2.1.2 Federal Process

The Government of Canada *Impact Assessment Act*, 2019 requires an Environmental Impact Assessment for federally designated projects. Projects that are not designated federally may still require screening in coordination with the provincial EA process.

Government agencies responsible for coordinating the EA processes include the BC Environmental Assessment Office (EAO) and the Impact Assessment Agency of Canada (IAAC). The two agencies have put in place an agreement to address when a project falls under both provincial and federal environmental assessment responsibility, at which time the two governments will cooperate on the EA while retaining their respective decision-making powers.

The Project will be assessed for whether it will require federal authorization by the Department of Fisheries and Oceans Canada (DFO) under paragraph 35(2)(b) of the *Fisheries Act* to commit harmful alteration, disruption, and destruction (HADD) of fish habitat or paragraph 34.4(2)(b) for any death of fish. An authorization would be required by the Minister, and habitat and/or productivity offsetting requirements acceptable to DFO and developed in consultation with participating Indigenous groups may need to be developed.

Whitebark pine and caribou are also valued components of federal legislative interest and will likely require specific studies and may require offset plans to be developed.

Additional environmental baseline survey work, including the TEM and determination of the extent of fish habitat, wetlands, and presence of listed species and ecosystems will provide information for federal regulators to consider in the EA process.

20.2.2 Mine Permitting

The Baptiste Nickel Project is in the pre-development stage and has only been subject to Notice of Work permitting through the BC Ministry of Energy, Mines and Low Carbon Innovation for seasonal surface exploration and geotechnical drilling programs. Anticipated permits required for the development and operation of the mine project include those listed in the following tables:

20.2.2.1 Provincial Authorizations

The following provincial authorizations are required or possible, in addition to and subsequent to the EAC.

Table 20-3: Possible Provincial Authorizations

Legislation	Authorization	Authorization Purpose
Drinking Water Protection Act	Water System Construction and Water System Operation	Construct and operate potable water supply system
Drinking Water Protection Act	Food Facility – Health Approval Application	Opening and operating a food service facility
Environmental Management Act	Effluent Discharge Permit	Authorizes discharges from a water treatment plant, sedimentation ponds, and a tails facility (seepage)
Environmental Management Act	Air Emissions Discharge Permit	Authorizes air emissions from incinerators and the process plant
Environmental Management Act	Solid Waste Discharge Permit	Authorizes disposal of solid waste into a landfill
Environmental Management Act	Hazardous Waste Registration	Authorizes a hazardous waste transfer facility
Environmental Management Act	Fuel Storage Registration	Authorizes a fuel storage facility
Environmental Management Act	Sewage Registration	Authorizes a sewage treatment plant
Forest Act	Occupant License to Cut	Authorizes harvesting of Crown timber
Forest Act	Road Use Permits	Authorizes use of Forest Service Roads
Forest and Range Practices Act	Special Use Permits	Authorizes road construction in a provincial forest
Heritage Conservation Act	Permits for Archaeological Impact Assessments	Authorizes site alteration and inspections
Land Act	License of Occupation	Authorizes occupancy and use of Crown land for e.g. transmission line or freshwater pipeline
Land Act	Temporary Use Permit	Temporary access road
Mineral Tenure Act	Mining Lease	Production of minerals
Land Title Act	Easements (e.g. transmission line)	Authorizes occupancy and use of private land
Mines Act	Permit for the Mine Plan and Reclamation Program	Approves the mine plan and reclamation program
Mines Act	Explosives Storage and Use Permit	Use, care, and transportation of explosives
Safety Standards Act	Permit	To connect to a powerline
Water Sustainability Act	License (Section 9/7)	Diversion, storage, or use of surface or groundwater for, e.g. storage dams, groundwater wells
Water Sustainability Act	Approvals/Notifications of changes in or about a stream (Section 11)	Changes in or about a stream for crossings associated with e.g. a waterline access road, transmission line
Wildlife Act	License	Designate no shooting area, relocation of wildlife during construction

20.2.2.2 Federal Authorizations

The Project will require Federal authorizations for the development of the mine project. Specific authorizations that may be required by the Federal government include:

Table 20-4: Possible Federal Authorizations

Legislation	Authorization	Authorization Purpose
Explosives Act	License	Manufacture, storage and use of explosives
Fisheries Act – Metal and Diamond Mine Effluent Regulations	Schedule 2 Amendment	Deposition of deleterious substances to fish bearing waterbodies
Fisheries Act (Section 34 and 35)	License	Death of Fish (Section 34) and Harmful, alteration, disruption of fish habitat (Section 35)
Migratory Birds Convention Act, (Section 5)	Permit	Authorizes periods during which migratory birds may be killed or captured and their nests or eggs destroyed
Nuclear Safety and Control Act	License	Radioisotope containing flow monitors
Radio Communications Act	License	Issuance and operation of designated frequency
Species at Risk Act (Section 32 and 33)	Permit	Provides for the protection of listed species in accordance with regulations
Transportation of Dangerous Goods	Permit	Transport of dangerous goods
Canadian Navigable Waters Act	Authorization	Authorizes work that may interfere with navigable waters

20.3 Indigenous Engagement

Local perspectives and early engagement are critical to FPX's decision-making processes throughout all aspects of the Project. Enduring relationships must be built with Indigenous Rightsholders, based on transparency, accountability, mutual understanding and respect for Indigenous rights and title, and a long-term commitment to shared value.

The Baptiste Deposit is located within the traditional territories of Tl'azt'en Nation and Binche Whut'en, and several Tl'azt'enne and Binche Whut'enne keyohs, a traditional governance system of the Dakelh people of the Stuart-Trembleur Lake area. First Nation territories that overlap the wider Decar mineral claims area include Takla Nation, Lake Babine Nation, and Yekooche First Nation.

FPX has maintained a high level of engagement with Indigenous communities over the last several years. Special attention has been paid to Indigenous engagement with the development of two key initiatives:

1. The Indigenous-led cultural and environmental baseline study programs. Study priorities have been guided by Indigenous knowledge and findings are being shared with the local communities on an ongoing basis.

2. The development and implementation of mine closure design workshops. The workshops are an opportunity for FPX and local Indigenous communities to discuss shared values and vision for long-term land use objectives..

20.4 Mine Design and Closure Planning

A conceptual mine closure and reclamation plan will be developed to support the EA process, and presented as a detailed plan as part of the joint Mines Act-Environmental Management Act permits Application. With approval of the project, and issuance of a Mines Act Permit, FPX will be required to review and if necessary, update the mine closure and reclamation plan at a minimum every five years through the life of mine.

20.4.1 Mine Waste and Water Management

As described in detail in Section 18, the proposed tailings facility is the key feature for the integrated waste and water management facilities that will be progressively developed and operated over the life of the mine. The tailings facility design considers management of mine waste in a single facility, utilizing open pit pre-stripping material and waste rock for dam construction.

The PFS tailings facility has been located in the upper reaches of a single catchment to reduce the quantity of contact water and to avoid fisheries sensitive watersheds of Sidney and Paula Creeks. Preliminary water balance modelling indicates the site to be in an annual water deficit, requiring allowance for freshwater makeup during operations. There is presently no expectation for active water treatment and discharge during mine construction and operations, owing to the predicted water deficit, however active water treatment has been considered for the active closure phase, when water will be discharged from the open pit.

FPX will continue to evaluate all aspects of the mine design, including alternatives for mine waste and water management, informed by future technical and environmental studies, and early engagement with Indigenous Rightsholders.

20.4.2 Closure and Reclamation Plan

The closure phase of the project will involve both active and long-term (post-closure) phases. Closure will be completed in a manner that will satisfy physical, chemical, and biological stability of the Project site. The reclamation and closure plan presented in this report is conceptual and based upon a general understanding of the geochemical considerations for the mine waste rock and tailings; additional evaluation of mine waste geochemistry, along with expected effects on water quality downstream of the various mine facilities will be required as design of the Project advances. Ecohydrological modelling will also need to be undertaken to inform reclamation prescriptions implemented to achieve mine closure objectives for ecosite re-establishment, biodiversity, and traditional and cultural uses.

The site will be progressively reclaimed towards the end of operations and at the end of mining. The primary objective of the closure and reclamation activities will be to return the site to a self-sustaining condition, in a form that respects the current watershed function. Activities that will be caried out to achieve these objectives will include:

- Overburden will be re-handled from the stockpile east of Sidney Creek for placement on the downstream face of the tailings facility dam. Overburden will be placed in a manner to create a ‘rough and loose’ surface that minimizes runoff and sediment transport. Reclamation of the tailings facility dams will begin as the final downstream toe is established and is expected to be complete prior to the end of mine operations.

- At the completion of tailings deposition within the tailings facility, overburden will be re-handled from the stockpile east of Sidney Creek for placement on the tailings beaches. The overburden cover over the tailings will be graded to limit high velocity runoff (through construction of small stilling ponds and meandering channels) and to isolate water on the surface of the facility.
- A closure spillway will be constructed in the left abutment of the North Dam, to manage extreme storm event runoff.
- Water from the tailings facility pond will be pumped to the open pit in active closure to accelerate development of an open pit lake. A spillway will be constructed at the south end of the open pit to route extreme storm event runoff to Lower Baptiste Lake.
- A water treatment plant will be installed in the area formerly occupied by the plant site as the open pit lake approaches the open pit spillway invert. This water treatment plant will treat water from the open pit and/or tailings facility pond until such time that seasonal outflows meet water quality objectives for the receiving environment.
- The various water management ponds around site will be removed when collected water meets the water quality objectives for the receiving environment.
- Mine site infrastructure (plant site, mining equipment, power line, etc.) will be removed when no longer required and footprints of disturbed areas will be reclaimed to limit runoff and sediment transport. Geotechnical instrumentation will be retained for use as long-term monitoring devices. Annual inspections of the site and ongoing evaluation of water quality, flow rates and instrumentation records will confirm design assumptions and performance of the closure activities.

It is anticipated that the major closure activities would be completed within a period of three years following the end of mining operations. Post-closure monitoring of reclamation success, geotechnical and chemical stability, will occur for several decades. Post-closure active or passive water treatment requirements will need to be determined. Total closure costs, including concurrent activities during operations, decommissioning, and the active and post-closure periods, are estimated at US$284 M. These costs have been included in the financial evaluation of the Project and costs associated with securing the reclamation and closure bond amount have also been considered and included in the owner's costs.

20.5 Next Steps

20.5.1 Cultural and Environmental Baseline Studies – Next Steps

FPX continues to undertake environmental studies and early engagement for the purpose of informing the Initial Project Description that will be used for consultation during the future Federal and Provincial EA process. To date, valued components have been identified by engagement with local communities by the Indigenous-led cultural and environmental consulting firm Shas-Ti Environmental. FPX understands that further critiques and definition of requirements for these studies will be accomplished during the initial review from both the Provincial and Federal environmental staff specialists. Modifications will be required throughout the assessment period to address concerns.

A key early engagement activity is the implementation of the mine closure workshop series. Initial workshops have been held on May 16 and July 5, 2023 and will continue throughout the development of the Project. FPX has also supported elder flights with a special emphasis on visiting Mt. Sydney Williams (Tselkun). These trips have provided an opportunity for the sharing of traditional knowledge and values within the Project area.

Programs to be initiated during 2023/2024 include a country foods assessment, in concert with advanced vegetation studies, a socio-economic analysis, as well as formal documentation of traditional and ecological knowledge in the region.

A hydrogeology and groundwater quality program, combined with surface hydrology and a water balance which is developed for the proposed mine facilities, will result in the development of a watershed model.

Linear features including the access road alignment and the power transmission line routes will have to undergo similar cultural and environmental baseline studies that have been commenced for the main mine project area. and as further definition of study requirements become available through the EAO and the Ministries responsible for administering the Lands Act permits required for development of these ancillary facilities.

These studies will support reviews during the environmental assessment process.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating cost estimates presented in this PFS provide substantiated costs that can be used to assess the economics of the Baptiste Nickel Project. The estimates are based on an open pit mining operation developed in two phases, a phased process plant, as well as associated tailings facility and infrastructure.

The estimates conform to Class 4 guidelines for a PFS-level estimate with a ±25% accuracy according to AACE International. Both estimates were developed in Q3 2023 US$ based on the proposed design for the Project, with input data from budgetary quotations for equipment, service contracts, and construction contracts; as well as Ausenco's in-house database of similar projects and studies, which includes experience from similar operations.

The capital and operating estimates were prepared or advised by the following groups:

- The mining was prepared by TechSer Mining Consultants Ltd. (TechSer).
- Process Plant, On-site Infrastructure, Consumables, and G&A were prepared by Ausenco.
- Off-Site Power was prepared by Carisbrooke Consulting Inc. (Carisbrooke).
- Tailings Facility was prepared by Knight Piésold Ltd. (KP).
- Off-site Access was prepared by Onsite Engineering Ltd. (Onsite Engineering).
- Owner's Cost and closure costs were prepared by FPX.

All cost amounts expressed are in US$ unless stated otherwise.

21.2 Capital Costs

21.2.1 Capital Cost Estimate Input

The following parameters and qualifications are considered in preparing the capital cost estimates:

- No allowance has been made for exchange rate fluctuations,
- There is no escalation added to the estimates.
- Growth allowances were included.

Data for the estimates have been obtained from numerous sources, including:

- scope of work;
- process design criteria;
- general arrangement drawings;

- drawings and sketches;
- process flow diagrams;
- single line diagrams;
- mechanical and electrical equipment lists;
- material take-offs;
- pre-engineering and modular building pricing;
- camp and services pricing;
- equipment pricing;
- reagent pricing; and
- contractor's cost data from recent similar projects.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs, and owner's costs) were identified and examined. Growth was allocated to each of these categories on a line-item basis based on the accuracy of the data. An overall contingency amount was derived in this fashion.

Vendors and contractors were requested to price in native currency. Pricing has been converted to US$ using the exchange rates in Table 21-1.

Table 21-1: Estimate Exchange Rates

CODE	Currency Description	Exchange Rate
USD	United States Dollar	1 US$ = 1.000 US$
AUD	Australian Dollar	1 US$ = 1.480 AUD
EUR	European Euro	1 US$ = 0.900 EUR
CAD	Canadian Dollar	1 US$ = 1.316 CAD

21.2.2 Capital Cost Estimate Summary

The estimate includes mining, processing, on-site infrastructure, tailings and waste rock facilities, off-site infrastructure, project indirect costs, project delivery, owner's costs, and contingency. The total capital cost summary is presented in Table 21-2. The total initial capital cost for the Baptiste Nickel project is US$2,182M; the Year 10 expansion capital cost is US$763M; and LOM sustaining costs are US$1,181M. Closure costs are estimated at US$284M.

Table 21-2: Summary of Total Capital Costs

WBS Level 1	WBS Description	Initial (US$M)	Expansion (US$M)	Sustaining (US$M)	Closure (US$M)	Total Cost (US$M)
1000	Mining	325	68	643	-	1,036
2000	Process Plant	730	379	0	-	1,109
3000	Tailings Facility	115	30	421	-	566
4000	On-Site Infrastructure	106	34	0	-	139
5000	Off-Site Infrastructure	127	1	0	-	128
	Subtotal Direct Costs	**1,403**	**511**	**1,064**	**-**	**2,978**
7000	Project Indirects	401	133	20	-	555
8000	Owner's Costs	106	16	0	-	122
9000	Contingency	272	103	97	-	471
Subtotal Indirect Costs		**779**	**252**	**117**	**-**	**1,148**
Project Total		**2,182**	**763**	**1,181**	**284**	**4,410**

Note: Values may not sum due to rounding.

21.2.3 Area 1000 – Direct Costs, Mining

The mining capital cost estimate is grouped into the following two main categories:

- Mine Infrastructure and Services – WBS 1100.
- Mining – WBS 1200.

The cost breakdown is shown in Table 21-3.

Table 21-3: Mining Capital Cost Estimate

Mining Capital Category	WBS	Initial (US$M)	Expansion (US$M)	Sustaining (US$M)	Total Cost (US$M)
Mine Infrastructure and Services	1100	33.9	-	-	33.9
Mining Services	1200	291.3	67.8	642.9	1,002.1
Total		**325.2**	**67.8**	**642.9**	**1,036.0**

Note: Values may not sum due to rounding.

21.2.3.1 Mine Infrastructure and Services

Mine infrastructure and services capital covers bulk earthworks and concrete in addition to the following buildings:

- mine office;
- mine dry;
- truck shop;
- truck wash;

- tire change;
- explosive/ magazine storage facility; and
- core shack.

21.2.3.2 Mining Services

Mine services capital costs include the following:

- Pioneering work:
  - Development of approximately 13 kilometers of mining haul roads.
  - Clearing, grubbing and topsoil removal.
  - Pads in mine phases 1 and 2 with sufficient operating space for mining haul trucks and excavators to operate.
  - Pad for the operational ore and overburden stockpile area.
- Pre-stripping: removal of approximately 38.6 Mt of waste and 14.8 Mt of ore to expose sufficient ore to allow continuous delivery to the concentrator upon start-up. These costs include all mine operating costs incurred before mill start-up, including grade control drilling, production drill and blasting, loading and hauling, support costs and supervision wages.
- Mine Equipment: mine equipment capital costs are derived from vendor quotations and operational data collected from other open pit mining operations. The distribution of mine equipment capital costs is based on the type and quantity of mobile equipment required for each year to meet mine production requirements. As new units are required, respective costs are included in that year. Fleet renewal requirements are based on defined operating life by equipment type, which are based on vendor input and data collected from other open pit mining operations. Minor equipment requirements were based on operational experience.
- Open pit dewatering: the system includes pumps and piping required to draw the existing groundwater level down below the active pit level and handle expected annual rainfall events. The pumps will be diesel and lift the water to the pit rim through a series of transfer pump stations. The final transfer pump station pumps to the process water pond, for reuse within the plant. The cost is carried under initial capital.
- The project capital costs also cover an allowance for the pit power loop, fuel station and explosives facility.

For further definition of mining services, see Section 16.

Table 21-3 summarizes the total mining capital cost estimate including a breakdown for initial, expansion and sustaining costs.

21.2.4 Area 2000 – Direct Costs, Process Plant

The definition of process equipment requirements was based on the process design criteria, process flowsheets and mass balance, as defined in Section 17. All major equipment was sized and included on a mechanical equipment list. Mechanical scopes of work, datasheets and schedules of pricing were developed and sent for budgetary pricing to equipment suppliers. 85% (Initial) and 86% (Expansion) of equipment on the mechanical equipment list was sourced from budgetary quotes; the remainder was priced from budget quotations or purchase orders from other recent estimates and projects.

Similarly, major electrical equipment was sized based on the project's electrical load list which was derived from the mechanical equipment list. Scopes of work, datasheets and schedules of pricing were developed to receive budgetary pricing from equipment suppliers. 47% (initial) and 58% (expansion) of equipment on the electrical equipment list was sourced from budgetary quotations. The remainder was priced from budget quotations or purchase orders from other recent estimates and projects.

The concrete quantities for the process plant pre-engineered building foundations were estimated based on 2D computer modelling and preliminary structural design. Ancillary buildings, the assay laboratory, camp and fuel storage and distribution were based on preliminary structural design only. All other quantities were scaled from the general arrangement drawings and benchmarked against Ausenco's historical data from reference projects.

Concrete take-offs for major structures including foundations, footings, walls, pedestals, slab on grade and elevated concrete, detailed excavation, and detailed backfill have been developed based on preliminary design calculations. Budget pricing was sourced on recent quotes from the market for supply, delivery, and installation of batched concrete. The concrete is inclusive of the following:

- The cost of batch plant mobilization and demobilization.
- The cost of supply and operation of batch plant including costs for the supply of cement and sand (aggregate to be provided by civil contractor) and winterization costs.
- The cost of materials includes formwork, required embedment and reinforcement steel.
- The cost for labour includes categorized installation hours multiplied by the productivity factor, direct labour rate and distributable rate based on recent contractor quotes.

Structural steel take-offs include medium designation and miscellaneous steel including grating, handrail and stair treads. It should be noted that the steel fabricators and installation contractors were not responsive to the request for current market quotations. Therefore, Ausenco carried unit rates and installation hours based on recent contractor quotes for projects in the region.

An allowance for the platework (chutes, bins, distribution boxes, launders, and shop-fabricated and field-erected tanks) has been developed by factoring the total installed mechanical by WBS level 3. A blended factor has been carried in the estimate and aligns with Ausenco's historical data and actual platework costs from the construction of recent projects.

An allowance for process plant piping (pipe, fittings, supports, valves, paint, special pipe items and flanges), instrumentation and remaining electrical bulks (cable trays, terminations, small lighting, and receptacles) has been developed by factoring the total installed mechanical by WBS level 3. A blended factor for each has been carried in the estimate and aligns with Ausenco's historical data and actual electrical bulk costs from construction of recent projects.

The process capital cost breakdown is shown in Table 21-4.

Table 21-4: Process Capital Cost Estimate

Process Capital Category	WBS	Initial (US$M)	Expansion (US$M)	Sustaining (US$M)	Total Cost (US$M)
Primary Crushing and Conveying	2100	57.0	29.2	-	86.2
Coarse Ore Stockpile and Reclaim	2200	33.6	19.6	-	53.2
Primary Grinding	2300	304.4	183.6	-	488.0
Magnetic Separation and Regrind	2400	125.6	75.0	-	200.6
Flotation	2500	86.9	38.8	-	125.7
Concentrate Handling	2600	33.7	12.8	-	46.5
Tailings Handling (excluding dam, tailings distribution and reclaim water systems)	2700	13.8	1.5	-	15.3
Reagents	2800	23.7	11.6	-	35.3
Process Plant Services and Common	2900	51.2	6.8	-	58.0
Total		**729.9**	**379.0**	**-**	**1,108.9**

Note: Values may not sum due to rounding.

21.2.5 Area 3000 – Direct Costs, Tailings Facility

Material take-offs related to the tailings facility were generated for site preparation, heavy civil construction, piping and mechanical systems. Material take-offs for each area were based on the following general assumptions/methodology:

- Site preparation and civil earthworks material take-offs were provided as neat, in-place quantities, with no allowance for swell, compaction, and waste.
- Overland piping for the water management systems were measured and provided as neat-line quantities without allowance for design growth, waste and snaking. Pipeline quantities were estimated as lengths by type of service with material, grade, and diameter specified.
- Major mechanical equipment (pumps and motors) was specified.

Prices for heavy civil construction are prepared using a combination of unit rates developed from first principles (i.e., equipment productivity, hourly equipment, and labour cost) along with KP judgement based on their experience. Budgetary quotations were solicited from vendors for larger mechanical equipment.

Electrical distribution to the tailings facility has been captured in WBS 4200.

Table 21-5: Tailings Facility Capital Cost Estimate

Tailings Facility Capital Category	WBS	Initial (US$M)	Expansion (US$M)	Sustaining (US$M)	Total Cost (US$M)
West Dam Construction	3100	74.4	-	120.5	194.8
North Dam Construction	3200	29.4	-	191.9	221.2
Tailings Distribution and Water Reclaim	3300	7.8	29.7	102.6	140.1
Water Collection and Diversion	3400	3.3	-	6.2	9.5
Total		**114.9**	**29.7**	**421.1**	**565.6**

Note: Values may not sum due to rounding.

21.2.6 Area 4000 – Direct Costs, On-site Infrastructure

The on-site infrastructure is grouped into the following main categories:

- Bulk earthworks – WBS 4100
- On-site power supply and distribution – WBS 4200
- Infrastructure buildings and facilities – WBS 4300
- Site water management – WBS 4400
- Permanent camp – WBS 4500
- Site services – WBS 4600
- Mobile equipment (non-mining) – WBS 4700

The on-site infrastructure cost breakdown is shown in Table 21-6.

Table 21-6: On-site Infrastructure Capital Cost Estimate

On-site Infrastructure Capital Category	WBS	Initial (US$M)	Expansion (US$M)	Sustaining (US$M)	Total Cost (US$M)
Bulk Earthworks	4100	26.3	9.7	-	36.0
On-site Power Supply and Distribution	4200	26.1	15.1	-	41.2
Infrastructure Buildings and Facilities	4300	20.2	-	-	20.2
Permanent Camp	4500	32.2	8.8	-	41
Site Services	4600	1.0	-	-	1.0
Mobile Equipment	4700	-	-	-	-
Total		**105.8**	**33.6**	**0.0**	**139.4**

Note: Values may not sum due to rounding.

21.2.6.1 Bulk Earthworks

Bulk earthworks for the process plant, infrastructure buildings, site roads and drainage, and freshwater and process water ponds were estimated from quantity take-offs from the Civil 3D software model and/or engineering calculations from 2D drawings (which come from the Civil 3D model). Sub-contract rates are used in the estimate for bulk earthworks requirements. Prices carried in the estimate are a combination of rates from a local contractor and Ausenco's historical rates.

Most bulk earthworks are completed for Phase 1 to provide sufficient footprint for: temporary construction facilities, plant site and mining infrastructure areas, access roads, haul roads, water management ponds and pipelines, truck-scale area, explosives facility, laydowns, on-site right-of-way (ROW) for 230kV power line and substation.

The expansion earthworks are for the excavation and haulage of the bulk material required for the conveyor corridor and second primary crusher ROM pad. The expansion plant location and expansion stockpile areas are already covered in the initial works and are not duplicated.

21.2.6.2 On-site Power Supply and Distribution

The new 230 kV substation on site will step down the 230 kV supply voltage to 34.5 kV, 4.16 kV, 600 V, and 120/208 V distribution, as required, for site-wide power distribution and will be located in close proximity to the concentrator. The site substation including associated electrical equipment is based on budgetary pricing from suppliers for initial and expansion phases.

Material take-offs for the overhead, on-site 34.5 kV powerline are based on the site plan drawings. Pricing is based on in-house engineering estimates ($/km) including design-supply-build delivery, civil works, and pole foundations.

21.2.6.3 Infrastructure Buildings and Facilities

The estimate includes the supply and installation of all the buildings within the process plant area and on-site infrastructure.

Building datasheets were developed to describe the requirements of each building, including sizing, load requirements, and features. Datasheets were included with the contract packages as a basis for pricing.

Pricing schedules were developed with a scope of work document, design criteria, and building layout drawings and sent to the market for budget pricing. The pre-engineered building quotes included supply, install, and delivery and the stockpile dome quotes included supply and delivery. Pricing includes supplying and delivering all materials, plant, equipment, tools, facilities, labour supervision, overhead, and all other items and services required to complete the works.

21.2.6.4 Permanent Camp

A 325-person permanent camp will be needed for Phase 1 operations. The operations will utilize the construction camp which will remain in place following construction. An additional 125 beds of the construction camp will be utilized by operations after the expansion is complete.

21.2.6.5 Site Services

Only the concrete for the on-site fuel storage facilities has been developed at this stage of the study. No other material take-offs have been developed for WBS 4600 – Site Services.

21.2.6.6 Mobile Equipment

The mobile equipment fleet is for supporting the ongoing operations of the process plant. It includes safety and maintenance vehicles as well as service equipment such as front-end loaders, cranes, skid-steers and forklifts.

The capital expenditure (CAPEX) is not included non-mining mobile equipment costs as they will be on a lease basis and so included in the operating cost estimate.

21.2.7 Area 5000 – Direct Costs, Off-site Infrastructure

The off-site infrastructure capital cost estimate is grouped into the following three main categories:

- Main Access Road – WBS 5100
- Water Supply – WBS 5200
- Power Supply – WBS 5300

The cost breakdown is shown in Table 21-7.

Table 21-7: Off-site Infrastructure Capital Cost Estimate

Off-site Infrastructure Capital Category	WBS	Initial (US$M)	Expansion (US$M)	Sustaining (US$M)	Total Cost (US$M)
Main Access Road	5100	72.0	-	-	72.0
Fresh Water Supply[1]	5200	1.4	0.5	-	1.9
Power Supply	5300	53.9	-	-	53.9
Total		**127.3**	**0.5**	**-**	**127.8**

Note: Values may not sum due to rounding. 1. The 14.3 km pipeline itself is covered under WBS 2930 (an additional US$26.4M initial cost)

21.2.7.1 Main Access Road

The existing forestry service roads (FSR) network currently used to access the site will be expanded and upgraded to allow unaffected year-round use. The Austin FSR in the immediate site area will be realigned with 16 km of new road construction to bypass the tailings facility. The overall access road length will be reduced by 39 km with the construction of the 7 km Middle River Bypass road segment, which includes a new Middle River bridge. The initial costs include all road upgrades and new road construction.

21.2.7.2 Water Supply

The fresh water supply for the site will be sourced from Trembleur Lake. WBS 5200 covers the initial and expansion pumps, both based on budgetary quotes, as well as earthworks associated with the pipeline. The 14.3 km pipeline itself is covered under WBS 2930 – Water Services (Process, Gland and Fresh).

21.2.7.3 Power Supply

Overhead off-site transmission powerline material take-offs and pricing were provided by Carisbrooke and have been included in the estimate. Firm pricing was used from similar projects using the same equipment as the basis of equipment supply pricing. Firm pricing construction proposals were used from similar projects located in British Columbia as the basis for installation costing. The pricing inputs were then applied to the project material take-offs. Direct costs include 155 km of 230 kV transmission line, including alignment clearing and access construction requirements.

21.2.8 Area 7000 to 9000 – Indirect Costs

The indirect costs include project indirect costs, owner's costs and contingency, as outlined below.

Project indirect costs include following:

- construction camp;
- miscellaneous distributable costs;
- commissioning representatives and assistance;
- spares;
- first fills and initial charges;
- freight and duties; and
- project delivery (EPCM).

Owner's costs include the following:

- Owner's team and expenses;
- operational readiness;
- environmental;
- land;
- communication;
- finance;
- HR; and
- administration.

Provision costs include contingency.

Indirect costs are summarized in Table 21-8 and are described in the following sections.

Table 21-8: Indirect Costs

Description	WBS	Initial(US$M)	Expansion(US$M)	Sustaining(US$M)	Total Cost (US$M)
Project Indirects	7000	401	133	20	555
Owner's Costs	8000	106	16	0	122
Contingency	9000	272	103	97	471
Total Indirect Capital Cost		**779**	**252**	**117**	**1,148**

Note: Values may not sum due to rounding.

21.2.8.1 Area 7000 – Project Indirects

21.2.8.1.1 Temporary Construction Facilities and Services (Project Preliminaries/Field Indirects)

Project preliminaries are items or services which are not directly attributable to the construction of specific physical facilities of plant or associated infrastructure but are required to be provided as support during the construction period.

These costs include:

- Temporary construction facilities - site offices, induction center, first aid facilities, administration, portable toilets, temporary fencing, temporary roads, and parking.
- Temporary utilities – power supply, temporary grounding and generators, construction lighting, and water supply.
- Construction support – site clean-up and waste disposal, material handling, maintenance of buildings and roads, testing and training, service labour, site transport, site surveys, QA/QC, and security.
- Construction equipment, tools, and supplies purchased by the owner or construction management (CM) contractor – heavy equipment and cranes, large tools, consumables, scaffolding, and purchased utilities.
- Material transportation and storage incurred by the owner or CM contractor – All types of freight, agents, staging, and marshalling.
- Site office – Local services and expenses, communications, and office furniture.

21.2.8.1.2 Construction Camp

The costs for establishing a construction camp facility, messing, and operations are included in the cost estimate.

The temporary camp is to be purchased at the beginning of the construction period (Initial phase), and the quantity of beds is calculated to cover all project requirements, including the Owner's supervision team.

The Project requires a 1,569-bed accommodation facility to support initial construction. The facilities include a 200-person early works camp followed by an additional 1,369 beds added prior to the peak of the construction period. To facilitate the construction of the expansion in Year 9, 749 of the initial phase construction beds will be utilized for the expansion construction crew.

Pricing schedules were developed with a scope of work document and pricing schedule and sent to the market for supply and installation budget quotations for the construction camp.

Pricing includes supply and delivery of all materials, plant, equipment, fuel, tools, facilities, labour supervision, overhead, and all other items and services required to complete the works, including fabricating and procuring all building components and carrying out all required inspections and testing. The returned pricing submissions were compared and evaluated.

21.2.8.1.3 Commissioning Representatives and Assistance

Commissioning assistance from mechanical completion to handover was developed to complement CM costs. In addition, a modification squad has been allowed for in the estimate. The modification squad has been carried out to assist the commissioning team to make minor modifications or provide labour assistance for commissioning.

Costs of vendor representatives for commissioning have been calculated from the mechanical and electrical equipment supply costs, 1.0% for construction vendor representatives and 1.5% for commissioning vendor representatives have been included in the cost estimate.

21.2.8.1.4 Spares

Major mechanical and electrical spares for commissioning purposes have been factored at 1.5% of the mechanical and electrical supply costs based on Ausenco’s historical data.

Major mechanical and electrical spares for operational purposes have been factored at 0.6% of the mechanical and electrical supply costs based on Ausenco’s historical data.

Major mechanical and electrical spares for capital/insurance purposes have been factored at 3.0% of the mechanical and electrical supply costs based on Ausenco’s historical data.

21.2.8.1.5 First Fills and Initial Charges

First fills include the costs for the initial construction first fills for installed equipment and process first fills, and consist of chemicals, fuels, and lubricants, etc.

First fills have been calculated from the mechanical and electrical equipment supply costs; total first fills equate to 1.0% for construction first fills and 1.5% for commissioning first fills.

21.2.8.1.6 Freight and Duties

The estimated freight costs have been determined by applying a percentage to the material and equipment supply costs by line item as part of the direct cost and were then relocated to the indirect cost category following FPX request. However, this relocation excludes commodities including architectural, earthworks, concrete, and the allowances for factored bulks (platework, process piping, electrical bulks, and Instrumentation) as the freight component is intrinsically built into the unit rates and so could not be extracted. Taxes and duties are excluded from the cost estimate.

21.2.8.1.7 Project Delivery (Engineering, Procurement, Construction, Management, EPCM)

EPCM services cost to cover such items as engineering and procurement services (home office-based), construction management services (site-based), project office facilities, information technology (IT), staff transfer expenses, secondary consultants, field inspection and expediting corporate overhead and fees.

The overall EPCM budget has been calculated as a percentage of the direct costs. Major cost categories covering the indirect costs are listed in the Table 21-9 by phase.

Table 21-9: Indirect Costs

Indirect Cost	Initial		Expansion		Sustaining	
	(US$M)	% of Direct Cost	(US$M)	% of Direct Cost	(US$M)	% of Direct Cost
Temporary Construction Facilities and Services	50.0	3.6	16.5	3.2	20.5	1.9
Construction Camp and Catering	119.2	8.5	14.1	2.8	-	-
Commissioning Reps and Assistance	2.2	0.2	1.3	0.2	-	-
Spares	14.7	1.0	6.5	1.3	-	-
First Fills and Initial Charges	5.6	0.4	3.2	0.6	-	-
Freight and Duties	49.9	3.6	22.9	4.5	-	-
Project Delivery	159.5	11.4	68.8	13.5	-	-
Total Indirect Cost	**401.1**		**133.3**		**20.5**	

Note: Values may not sum due to rounding.

21.2.8.2 Area 8000 – Owner's Costs

Owner's costs were estimated by FPX through a combination of first principles build-up and benchmarking. Owner's costs include the owner's team and expenses, operational readiness, environmental, land, communication, finance, human resources, and administration. Table 21-10 presents Owner's Costs by phase.

Table 21-10: Owner's Costs

	Initial (US$M)	Expansion (US$M)	Sustaining (US$M)	Total Cost (US$M)
Owner's Costs	106.0	16.0	-	122.0

21.2.8.2.1 Closure Costs

FPX developed an order of magnitude (OOM) estimate for the closure requirement. This included the tailings facility closure and a water treatment plant in addition to all necessary demolition, rehabilitation, revegetation, earth grading/contouring, scrap metal disposal/tipping fees, and post-closure monitoring. The total closure cost was calculated to be US$284 M.

21.2.8.3 Area 9000 – Contingency

Contingency is a provision of funds for unforeseen or inestimable costs within the defined project scope relating to the level of engineering effort undertaken and estimate/engineering accuracy and applied to provide an overall level of confidence in costs and schedule outcomes. The contingency is meant to cover events or incidents that occur during the course of the project that cannot be quantified during the estimate preparation and does not include any allowance for project risk.

It is important to note that contingency does not cover scope changes, force majeure, adverse weather conditions, changes in government policies, currency fluctuations, escalation, and other project risks.

A summary of the contingency by WBS is noted below in Table 21-11.

Table 21-11: Contingency by WBS

Description	WBS	Initial (US$M)	Expansion (US$M)	Sustaining (US$M)	Total (US$M)
Mining	1000	15.1	4.1	32.1	51.3
Process Plant	2000	126.5	65.8	0	192.3
Tailings Facility	3000	17.4	4.6	61.5	83.5
On Site Infrastructure	4000	20.1	6.3	0	26.4
Off Site Infrastructure	5000	19.3	0.1	0	19.4
Project Indirects	7000	62.5	20.6	3	86.1
Owner's Costs	8000	10.6	1.6	0	12.2
Total		**271.5**	**103.1**	**96.6**	**471.2**

Note: Values may not sum due to rounding.

21.2.8.3.1 Growth Allowance

Each line item of the estimate is developed initially as a bare quantity and cost. A growth allowance has then been allocated to each element of direct cost line items to reflect the level of definition in design (quantity maturity) and pricing strategy (cost maturity).

Estimate growth is:

- Intended to account for items that cannot be quantified based on the current engineering status but are empirically known to appear, essentially bridging the gap from study to constructed quantities/costs.
- Accuracy of quantity take-offs and engineering lists based on the level of engineering and design undertaken at a prefeasibility study level.
- Pricing growth for the likely increase in cost due to the development and refinement of specifications as well as re-pricing after initial budget quotations and after finalization of commercial terms and conditions to be used on the Project.

Where an allowance has been used which is the result of factoring, no growth has been applied as the factor has been surmised from an actual cost.

Growth has been calculated at the line-item level by evaluating the status of the engineering scope definition and maturity and the ratio of the various pricing sources for equipment and materials used to compile the estimate as detailed in Table 21-12.

Table 21-12: Total Growth Allowances

Description	WBS	Initial		Expansion		Sustaining	
		($M)	% Growth Applied	($M)	% Growth Applied	($M)	% Growth Applied
Mining	1000	5.0	2	0.6	1	-	-
Process Plant	2000	45.8	7	23.5	7	-	-
Tailings Facility	3000	9.4	9	2.5	9	38.3	10
On-site Infrastructure	4000	10.9	12	3.0	10	-	-
Off-site Infrastructure	5000	8.3	7	0.0	6	-	-
Total		**79.5**		**29.7**		**38.3**	

Note: Values may not sum due to rounding.

21.2.9 Exclusions

The following items are not considered in the PFS capital cost estimate:

- operating costs;
- taxes and duties;
- permitting;
- scope changes;
- special incentives (schedule, safety or others);
- escalation;
- environmental impact assessment;
- lost time due to weather, labour availability and disruption or force majeure events; and
- arctic corridors for access between buildings.

21.3 Operating Costs

21.3.1 Operating Cost Estimate Summary

The operating cost estimate for the project covers Year 1 of production through to the end of mine life and is detailed in the subsequent subsections. Two distinctive phases of the processing plant include:

1. Phase 1 (Years 1 to 9): the process plant is operated at a throughput of 108 kt/d.

2. Phase 2 (Years 10-29): the process plant goes through an expansion to accommodate a throughput of 162 kt/d.

Operating costs include the ongoing cost of operations relating to mining, processing, general and administrative, and product transportation costs. Table 21-13 to Note: Values may not sum due to rounding.

Table 21-16 provides a breakdown of the operating costs into individual cost centers for each phase and LOM of the Project.

Table 21-13: Life-Of-Mine Operating Cost and AISC

Category	Units	Phase 1	Phase 2	LOM Average
Mining	US$/t milled	2.59	3.31	3.14
Processing	US$/t milled	3.75	3.59	3.63
G&A	US$/t milled	1.23	1.05	1.09
Concentrate Transport	US$/t milled	0.31	0.29	0.29
Total Cash Costs	US$/t milled	7.88	8.24	8.15
C1 Operating Cost[1]	US$/lb nickel produced	3.48	3.76	3.70
AISC[2]	US$/lb nickel produced	3.97	4.23	4.17

Notes: 1. Exclusive of any byproduct credits. 2. Inclusive of operating cost, expansion capital, sustaining capital, royalties, and closure capital.

Table 21-14: Operating Cost Summary – Phase 1 Average

Cost Area	Annual Total (US$M/a)	US$/t Milled	US$/lb Ni	% of Total
Mining	99	2.59	0.98	32.9
Process	144	3.75	1.42	47.6
G&A	47	1.23	0.47	15.6
Concentrate Transport	12	0.31	0.12	3.9
Total	**302**	**7.88**	**2.99**	**100**

Note: Values may not sum due to rounding.

Table 21-15: Operating Cost Summary – Phase 2 Average

Cost Area	Annual Total (US$M/a)	US$/t Milled	US$/lb Ni	% of Total
Mining	195	3.31	1.32	40.2
Process	211	3.59	1.43	43.6
G&A	62	1.05	0.42	12.7
Concentrate Transport	17	0.29	0.12	3.5
Total	**484**	**8.24**	**3.28**	**100**

Note: Values may not sum due to rounding.

Table 21-16: Operating Cost Summary – LOM Average

Cost Area	Annual Total (US$M/a)	US$/t Milled	US$/lb Ni	US% of Total
Mining	164	3.14	1.24	38.5
Process	190	3.63	1.43	44.5
G&A	57	1.09	0.43	13.3
Concentrate Transport	15	0.29	0.12	3.6
Total	**427**	**8.15**	**3.21**	**100**

Note: Values may not sum due to rounding.

21.3.2 Basis of Estimate

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q3 2023 pricing.
- All figures are expressed in USD and consider the exchange rates detailed in Table 21-1.
- Labour is primarily sourced locally and assumed labor rates are typical of the region.
- Worker rotations will be a 2-week-on/2-week-off schedule. Regional workers will utilize a FIFO structure to Prince George and transfer to a bus which will then take them to site. Local workers will rely on the bus to travel between site and designated muster points.
- Electrical power demand and consumption charge is based on BC Hydro's published rates at US$0.039/kWh and US$6.67/kVa/month.
- Steel media consumption rates have been estimated based on the mill power draw required and average material hardness.
- Reagent consumption rates have been estimated based on metallurgical test work and standard operating practices and are costed using a blend of historical and supplier quotes. All reagent and consumable costs include transportation to site.
- Diesel price of US$0.90/L.
- Equipment liners and consumables are based on vendor quotations.
- Light vehicles and other non-mining mobile equipment are estimated on a leased basis and included in the operating cost estimate.

21.3.3 Mine Operating Costs

Mining costs were estimated on a first principles basis for each operating year, and results in a LOM average cost of US$3.14/t milled. Phase 1 mining costs average US$2.59/t milled, while Phase 2 mining costs average US$3.31/t milled. The lower Phase 1 mining costs are due to the lower strip ratio in early operating years.

Mining costs include:

- Fuel and power: based on a listing of required equipment and vendor suggested consumption rates.
- Consumables: includes tires, replacement parts, lubricants, and ground engagement tools.
- Salaries and wages: based on an estimate of staff and labour requirements and regional labour rates.
- Costs have been included for contracting blasting material supply and services.

Table 21-17: Mining Operating Costs

Units Costs	Phase 1		Phase 2		LOM	
	US$/t Milled	US$/t Mined	US$/t Milled	US$/t Mined	US$/t Milled	US$/t Mined
Drilling	0.15	0.12	0.19	0.12	0.18	0.12
Blasting	0.42	0.29	0.42	0.26	0.42	0.27
Loading	0.34	0.24	0.34	0.22	0.34	0.22
Hauling	1.10	0.77	1.88	1.18	1.70	1.09
Aux Eq. and Road Maintenance	0.28	0.21	0.29	0.18	0.29	0.19
Supervision Wages	0.29	0.23	0.19	0.12	0.21	0.14
Total	**2.59**	**1.86**	**3.31**	**2.08**	**3.14**	**2.03**

Note: Values may not sum due to rounding.

21.3.4 Process Operating Costs

21.3.4.1 Labour

Labour will be primarily sourced from the local community, such as Tachie, Binche, Fort St. James, Vanderhoof, and Prince George. It is expected 65% of the total workforce will be composed of local labour. Local labourers will have bus access to and from site at the start and end of their work rotation. Personnel coming from outside of local communities will fly to Prince George and bus to site.

Staffing was estimated by benchmarking against other similar local projects. Labour costs include all required personnel for the process operating, maintenance, laboratory, administrative personnel, and site services. The initial and expansion phase of the mill has an on-roll total headcount of 175 and 272, respectively, and includes metallurgy, mill operations, tailings operations, maintenance, and laboratory. A payroll burden of 35% is carried to account for benefits and other non-salary labour elements. Table 21-18 breaks down the processing labour costs by department for each phase of the project.

Table 21-18: Processing Labour by Department

Role	Phase 1 (headcount)	Phase 2 (headcount)
Operations – Processing	73	124
Operations – Maintenance	65	95
Tailings	15	21
Laboratory	22	32
Total	**175**	**272**

Table 21-19: Processing Labour Operating Cost

Labour	Phase 1		Phase 2		LOM Average	
	Total (US$M/a)	US$/t Milled	Total (US$M/a)	US$/t Milled	Total (US$M/a)	US$/t Milled
Mill Operations	8.9	0.23	14.8	0.25	12.9	0.25
Maintenance	8.6	0.22	12.4	0.21	11.2	0.21
Tailings	1.8	0.05	2.4	0.04	3.2	0.06
Laboratory	2.5	0.07	3.6	0.06	2.2	0.04
Total	**21.7**	**0.57**	**33.1**	**0.56**	**29.5**	**0.56**

Note: Values may not sum due to rounding.

21.3.4.2 Reagents and Consumables

The reagent and consumables used and associated consumption rates are summarized in Section 17. Reagent prices were sourced from local distributors at bulk rates and include freight cost to site. For all consumables other than reagents, a domestic freight factor of 10% is applied to the cost of the item. For international shipping from the US and overseas, 12% and 14% are considered, respectively.

Table 21-20: Consumables and Reagents

Item	Phase 1		Phase 2		LOM Average	
	US$M/a	US$/t Milled	US$M/a	US$/t Milled	US$M/a	US$/t Milled
Reagents	39.8	1.04	53.3	0.91	49.0	0.94
Consumables	20.6	0.54	30.7	0.52	27.5	0.53
Total	**60.4**	**1.58**	**84.2**	**1.43**	**76.7**	**1.46**

Note: Values may not sum due to rounding.

21.3.4.3 Maintenance Parts and Supplies

Maintenance consumable costs (Table 21-21) were derived for each phase of the project by applying factors to the total installed costs of the mechanical and electrical equipment. A factor of 5% was used for the crushing circuit and 3% was used for the remaining concentrator, tailings, and on-site infrastructure areas. The factors were determined by benchmarking maintenance costs derived from nearby projects with similar operating parameters.

Table 21-21: Maintenance Consumable Costs

Description	Phase 1		Phase 2		LOM Average	
	US$M/a	US$/t Milled	US$M/a	US$/t Milled	US$M/a	US$/t Milled
Crushing	1.1	0.03	1.6	0.03	1.5	0.03
Process	6.0	0.16	9.4	0.16	8.3	0.16
Tailings	1.1	0.03	1.7	0.03	1.5	0.03
On-site Infrastructure	0.1	0.01	0.1	0.00	0.01	0.01
Total	**8.2**	**0.21**	**12.9**	**0.22**	**11.4**	**0.22**

Note: Values may not sum due to rounding.

21.3.4.4 Power

Electrical power cost was derived from the BC Hydro published rates for Q2 2023. For greater than 60 kV supply, a consumption charge of US$0.0391/kWh and a base demand charge of US$6.67/kVa/month is billed monthly.

The average power draw was determined by determining the required amount of power per piece of refinery equipment and factoring the motor efficiency.

21.3.4.5 Mobile Equipment

Vehicle costs are based on a scheduled number of light vehicles and other non-mining mobile equipment. Included in the costs are fuel, maintenance, spares, consumables, and cost of leasing. For diesel consuming vehicles and equipment, a 3% diesel exhaust fluid factor was placed on the total diesel consumption. Annual mobile equipment cost for Phase 1 is $0.6M per annum and $0.8M per annum for Phase 2.

21.3.5 General and Administrative Operating Costs

General and Administrative (G&A) costs are expenses not directly related to the production of the mill concentrate and are expenses that are not covered under mining, processing, or concentrate transport costs. The G&A costs were developed using a combination of Ausenco's in-house data on existing operations, publicly available benchmark data, and inputs from FPX.

G&A operating costs were categorized into five major cost centers: labour, vehicles, travel and camp, surface operations, and owner's costs. Owner's costs include safety, environmental, administrative, and general costs.

Table 21-22: G&A Operating Costs

Department Area	Phase 1		Phase 2		LOM Average	
	$M/a	$/t Milled	$M/a	$/t Milled	$M/a	$/t Milled
Labour	8.9	0.23	9.9	0.17	9.6	0.18
Vehicles	0.6	0.02	0.7	0.01	0.6	0.01
Travel and Camp	10.2	0.27	16.4	0.28	14.5	0.28
Surface Operations	1.3	0.03	1.9	0.03	1.7	0.03
Owner's Costs	26.3	0.69	32.6	0.55	30.6	0.58
Total	**47.2**	**1.23**	**61.5**	**1.05**	**56.9**	**1.09**

Note: Values may not sum due to rounding.

21.3.6 Concentrate Handling and Transport

The costs associated with concentrate handling and transport outside of the plant site have been incorporated into the operating costs above. The rate considers transport from Baptiste to Asia via trucking to Prince George, Rail to Prince Rupert and then ocean freight to South Korea. Third-party shipping companies are assumed for the rail transport, offload/load onto ship, demurrage, detention, storage and documentation fees, ocean freight to Asia, terminal handling fees in Asia (including container cleaning/maintenance) and tax import fees.

The total cost is $120.44/t Shipped and $0.29/t Milled (LOM average). Note: cost/t Milled has minimal variation for each phase from the LOM average, however the detail is presented in Table 21-14 to Table 21-16.

21.3.7 Owner's Costs

The owner's cost center covers administrative, general, safety, and environmental costs. These were developed by FPX through a combination of first principles and benchmarking off comparable large-scale BC mines. See Table 21-23 for a breakdown of the owner's costs by phase.

Table 21-23: Owner's Cost

Department Area	Phase 1		Phase 2		LOM Average	
	US$M/a	US$/t Milled	US$M/a	US$/t Milled	US$M/a	US$/t Milled
Administration	9.5	0.25	11.9	0.20	11.1	0.21
General	9.0	0.23	10.8	0.18	10.2	0.20
Safety	3.0	0.08	3.8	0.07	3.6	0.07
Environmental	4.8	0.12	6.1	0.10	5.7	0.11
Total	**26.3**	**0.69**	**32.6**	**0.55**	**30.6**	**0.58**

Note: Values may not sum due to rounding.

A summary of items included in each owner's cost includes:

- Administration:
  - travel and entertainment;
  - regulatory compliance;
  - insurance;
  - land costs;
  - legal;
  - FSJ Office - Rent/Utilities/Supplies;
  - tax and audit;
  - recruiting/relocation;
  - community contributions (non-IBA);
  - public relations; and
  - corporate overhead.

- General:
    - IT - hardware and software;
    - consulting services;
    - operating supplies;
    - communication;
    - janitorial;
    - access road maintenance;
    - power line maintenance; and
    - security services and supplies.
- Safety:
    - fire protection services;
    - safety training;
    - emergency medical services;
    - paramedical services; and
    - safety supplies;
- Environmental:
    - permits and fees;
    - monitoring;
    - hazardous waste program;
    - supplies;
    - analytical services; and
    - service contracts.

21.3.8 Closure and Reclamation Planning

The closure and reclamation costs for the project consider:

- Annual costs for closure and reclamation bonding.
- Concurrent reclamation (Years 1-29): occurs throughout the LOM and is completed in areas no longer actively utilized by mine operations, such as revegetation of final downstream embankment slopes.
- Decommissioning (Year 30-31): decommissioning and demolition of process facilities and footprints of disturbed areas reclaimed to limit runoff and sediment transport, hazardous waste removal, decommissioning and regrading of the open pit, and decommissioning of the tailings discharge and reclaim water systems including removal of pumps and pipelines, and decommissioning of the transmission line.

- Active Closure (Years 30-35): site regrading and revegetation, tailings facility dam downstream face overburden cover placement, tailings facility beach overburden cover placement and emergency spillway construction, open pit emergency spillway construction, removal of water management ponds, active water treatment of site contact water prior to discharge to the environment, ongoing environmental monitoring.
- Post-Closure (Years 35+): geotechnical instrumentation retained for long-term monitoring, annual site inspections and ongoing evaluation of water quality, flow rates and instrumentation to confirm performance of closure activities.

The total costs associated with closure are approximately US$284 M.

22 ECONOMIC ANALYSIS

22.1 Forward-Looking Information Cautionary Statements

The results of the economic analyses discussed in this section represent forward- looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes:

- mineral resource and reserve estimates;
- assumed commodity prices and exchange rates;
- the proposed mine production plan;
- projected mining and metallurgical recovery rates;
- sustaining costs and proposed operating costs;
- assumptions as to closure costs and closure requirements; and
- assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- changes to costs of production from what are estimated;
- unrecognized environmental risks;
- unanticipated reclamation expenses;
- unexpected variations in quantity of mineralized material, grade, or recovery rates;
- geotechnical or hydrogeological considerations during mining being different from what was assumed;
- failure of mining methods to operate as anticipated;
- failure of plant, equipment, or processes to operate as anticipated;
- changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis;
- ability to maintain the social licence to operate;
- accidents, labour disputes and other risks of the mining industry;
- changes to interest rates;
- changes to tax rates; and
- deviations in commodity prices from those assumed in the financial model.

22.2 Methodologies Used

An engineering economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the project based on an 8% discount rate. A detailed tax build-up was conducted by a third-party tax expert, however tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations. Sensitivity analysis was performed to assess the impact of variations in discount rate, nickel price, metallurgical recovery, operating costs, and initial capital costs. The capital and operating cost estimates were developed specifically for this project and are summarized in Section 21 of this report (presented in Q3 2023 US$). All presented figures are in US$ unless otherwise noted.

22.3 Financial Model Parameters

The economic analysis was performed using the following assumptions:

- construction period of 36 months;
- mine life of 29 years;
- a long-term nickel price assumption of US$19,290/t (US$8.75/lb) is assumed, which is consistent with the average long-term nickel price forecast published by six base metals analysts, as presented in Section 19. The price used is meant to reflect the average nickel price expectation over the life of the project. No price inflation or escalation factors were taken into account;
- United States to Canadian dollar exchange rate assumption of 1.00 CAD : 0.76 US$;
- cost estimates in constant 2023 US$ with no inflation or escalation factors considered;
- no salvage value assumed;
- results are based on 100% ownership;
- a 1% NSR is considered, applicable to revenue less concentrate transport cost;
- capital costs funded with 100% equity (i.e., no financing costs assumed);
- all cash flows discounted to beginning of construction;
- all metal products are assumed sold in the same year they are produced; and
- project revenue is derived from the sale of awaruite concentrate briquettes and MHP.

22.3.1 Taxes

The Project has been evaluated on an after-tax basis to provide approximate value of the potential economics. The tax model was prepared by an independent tax consultant. As of the effective date of this report, the project was assumed to be subject to the following tax regime:

- The Canadian Corporate Income Tax system consists of the federal income tax (15%) and the provincial income tax (12%).
- The BC Mineral Tax was modelled using a net current proceeds rate of 2% and a net revenue tax rate of 13%.

22.3.1.1 Clean Technology Manufacturing Investment Tax Credit

The tax model also includes the application of a clean technology manufacturing investment tax credit (CTMITC). Legislation regarding the CTMITC has not yet been presented, but the Federal government has announced that it will be a 30% refundable investment tax credit (ITC). It is assumed that, as with other refundable ITCs, the CRA will follow a protocol of auditing claims before paying any refund, and that refunds will be paid 12-18 months following the due date for filing the claim (i.e. in the second taxation year following the taxation year in which the claim arises). For income tax purposes, the amount of the ITC deducted from the undepreciated capital cost of the assets in the year following the year of claim. It is assumed that all depreciable capital costs associated with the project will qualify for the CTMITC, except for access road costs and owner's costs.

22.3.1.2 BC Mineral Tax

With regards to the BC Mineral Tax, the following assumptions have been made:

- The amount of the CTMITC is deducted from the cumulative expenditure account (CEA) in the year of the ITC claim (the year it becomes receivable).
- The New Mine Allowance is in effect for new or expanded mines where production of minerals commences before December 31, 2025. Historically, new mine allowances have been reintroduced before the program has concluded. It is assumed that the new mine allowance will continue to be available after December 31, 2025.
- The tax model assumes a 3.75% imputed interest rate used in determining the investment allowance for BC Mineral Tax purposes. This percentage is calculated as an average of the monthly Bank of Canada (BOC) rate, multiplied by 1.25, and again multiplied by the days in the filing period (normally 365/365). While the current BOC rate is 5%, 3% is an historically reasonable LOM rate.

22.3.2 Working Capital

Working capital assumptions include accounts receivable (0 days), inventories (30 days), and accounts payable (30 days). The effective sum of working capital over the LOM is zero.

22.3.3 Royalty

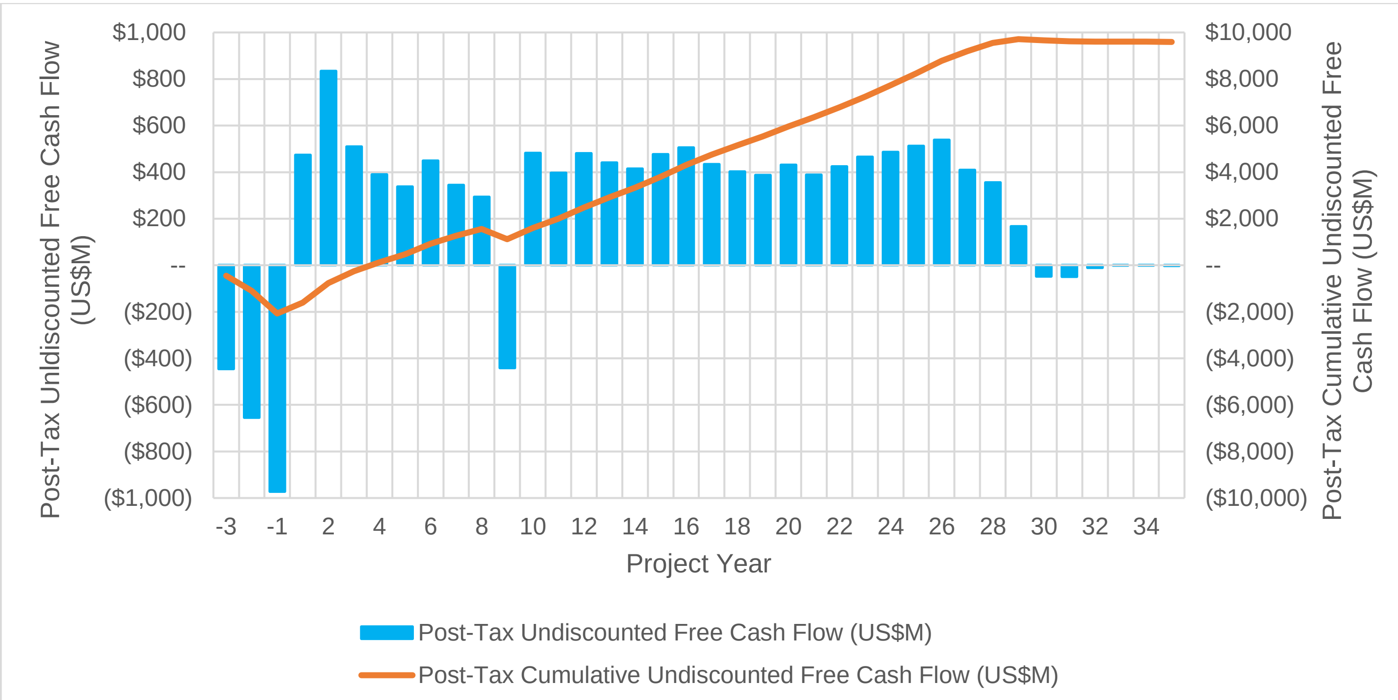
The project has a 1% NSR royalty, applicable to revenue less concentrate transport cost. This results in approximately US$308 million in royalty payments over the LOM.

22.4 Economic Analysis

The economic analysis was performed assuming an 8% discount rate. The pre-tax net present value (NPV) discounted at 8% is US$2,923 M, the IRR is 19.1%, and payback is 4.6 years. On an after-tax basis, the NPV 8% is US$2,010 M, the IRR is 18.6%, and the payback period is 3.7 years. The after-tax payback period is shorter than the pre-tax payback period due to the inclusion of investment tax credit refunds in Year -1 to Year 2 of the Project.

A summary of the project economics is shown graphically in Figure 22-1 and included in Table 22-1. The cashflow on an annualized basis is provided in Table 22-2.

Figure 22-1: Projected Post-Tax Undiscounted Cash Flow



Source: Ausenco, 2023.

Table 22-1: Summary of Project LOM Cashflow Assumptions and Results

General	Unit	LOM Total / Average
Nickel Price	US$/lb	8.75
Exchange Rate	C$:US$	0.76
Discount Rate	%	8
Mine Life	years	29
Total Waste Tonnes Mined (excluding capitalized pre-stripping)	kt	837,576
Total Mill Feed Tonnes	kt	1,487,721
Strip Ratio (excluding capitalized pre-stripping)	waste:ore	0.56
Production	**Unit**	**LOM Total / Average**
Concentrator Head Grade, DTR Ni	%	0.13
DTR Ni Recovery – Awaruite Concentrate	%	82.2
DTR Ni Recovery – MHP	%	6.5
Total DTR Ni Recovery (Awaruite Concentrate + MHP)	%	88.7
Nickel Produced – Awaruite Concentrate	mlbs	3,505
Nickel Produced – MHP	mlbs	277
Total Nickel Produced (Awaruite Concentrate + MHP)	mlbs	3,781
Total Average Annual Nickel Production (LOM)	mlbs	130
Total Average Annual Nickel Production (LOM)	kt	59.1
Nickel Payability – Awaruite Concentrate	%	95
Nickel Payability – MHP	%	87
Total Payable Nickel	mlbs	3,570
Total Payable Nickel	Mt	1.62
Operating Costs	**Unit**	**LOM Total / Average**
Mining	US$/t milled	3.14
Processing	US$/t milled	3.63
General and Administration	US$/t milled	1.09
Concentrate Transport	US$/t milled	0.29
Total Operating Cost	US$/t milled	8.15
Royalties	**Unit**	**LOM Total / Average**
NSR	%	1
C1 and AISC	**Unit**	**LOM Total / Average**
C1 Cost*	US$/lb Ni	3.70
All-in Sustaining Cost (AISC)**	US$/lb Ni	4.17
Capital Costs	**Unit**	**LOM Total / Average**
Initial	US$M	2,182
Sustaining	US$M	1,181
Growth	US$M	763
Closure	US$M	284
Total Capital Costs	US$M	4,410
Financials - Pre-Tax	**Unit**	**LOM Total / Average**
Pre-Tax $NPV_{8\%}$	US$M	2,923
Pre-Tax IRR	%	19.1
Pre-Tax Payback	years	4.6
Financials - Post-Tax	**Unit**	**LOM Total / Average**
Post-Tax $NPV_{8\%}$	US$M	2,010
Post-Tax IRR	%	18.6
Post-Tax Payback	years	3.7

* C1 costs consist of operating cash costs (which includes transportation charges) plus treatment and refining charges.

** AISC consist of total cash costs (C1 costs) plus royalties, sustaining capital, BC hydro integration, and closure cost.

Table 22-2: Projected LOM Post-Tax Unlevered Free Cash Flow

Dollar figures in Real Q3 2023 US$M unless otherwise noted		Project Year																			
	Unit	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**
Macro Assumptions																					
Nickel Price	US$/lb	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75
Exchange Rate	C$:US$	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76
Free Cash Flow Valuation																					
Revenue	**US$M**	**--**	**--**	**--**	**659**	**866**	**918**	**868**	**869**	**932**	**879**	**751**	**755**	**1,079**	**1,297**	**1,341**	**1,181**	**1,237**	**1,293**	**1,333**	**1,223**
Operating Costs	US$M	--	--	--	(274)	(290)	(308)	(312)	(312)	(307)	(312)	(292)	(308)	(436)	(506)	(516)	(485)	(535)	(528)	(507)	(504)
Royalties	US$M	--	--	--	(6)	(9)	(9)	(9)	(9)	(9)	(9)	(7)	(7)	(11)	(13)	(13)	(12)	(12)	(13)	(13)	(12)
EBITDA	**US$M**	**--**	**--**	**--**	**378**	**567**	**600**	**548**	**548**	**616**	**558**	**452**	**440**	**633**	**778**	**812**	**684**	**690**	**753**	**813**	**708**
Initial Capital	US$M	(436)	(654)	(1,091)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Sustaining Capital	US$M	--	--	--	(74)	(16)	(38)	(41)	(87)	(11)	(34)	(12)	(38)	(12)	(159)	(85)	(32)	(57)	(28)	(33)	(35)
BC Hydro Integration	US$M	(9)	--	--	1	1	1	1	1	1	1	1	--	--	--	--	--	--	--	--	--
Expansion Capital	US$M	--	--	--	--	--	--	--	--	--	--	--	(763)	--	--	--	--	--	--	--	--
Closure Costs	US$M	--	--	--	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)
Changes in Working Capital	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Pre-Tax Unlevered Free Cash Flow	**US$M**	**(446)**	**(654)**	**(1,091)**	**303**	**549**	**561**	**505**	**459**	**603**	**523**	**439**	**(363)**	**618**	**616**	**723**	**650**	**630**	**722**	**777**	**670**
Pre-Tax Cumulative Unlevered Free Cash Flow	*US$M*	*(446)*	*(1,100)*	*(2,191)*	*(1,888)*	*(1,339)*	*(778)*	*(273)*	*186*	*790*	*1,312*	*1,751*	*1,388*	*2,006*	*2,622*	*3,345*	*3,995*	*4,624*	*5,347*	*6,124*	*6,794*
Tax Payable	**US$M**	**--**	**--**	**(118)**	**(171)**	**(285)**	**52**	**115**	**122**	**154**	**178**	**146**	**77**	**137**	**220**	**243**	**209**	**215**	**246**	**272**	**236**
Pre-Tax Unlevered Free Cash Flow	**US$M**	**(446)**	**(654)**	**(972)**	**474**	**834**	**509**	**390**	**337**	**449**	**344**	**293**	**(441)**	**481**	**396**	**480**	**441**	**415**	**476**	**505**	**433**
Pre-Tax Cumulative Unlevered Free Cash Flow	*US$M*	*(446)*	*(1,100)*	*(2,073)*	*(1,599)*	*(765)*	*(256)*	*134*	*471*	*920*	*1,264*	*1,557*	*1,117*	*1,598*	*1,994*	*2,475*	*2,915*	*3,330*	*3,807*	*4,312*	*4,745*
Production																					
Mill Feed	kt	--	--	--	29,565	39,420	39,420	39,420	39,420	39,420	39,420	39,420	39,342	52,786	59,130	55,256	49,102	58,089	59,130	59,130	59,130
Ni Grade	%	--	--	--	0.14	0.14	0.14	0.14	0.14	0.15	0.14	0.12	0.12	0.13	0.14	0.14	0.12	0.13	0.14	0.14	0.13
Ni Recovery - Awaruite Concentrate	%	--	--	--	81.2	81.1	81.6	81.5	81.5	82.3	82.3	81.7	81.2	81.2	82.7	83.0	81.2	81.4	81.0	81.8	81.7
Ni Recovery - MHP	%	--	--	--	7.0	7.0	6.8	6.8	6.8	6.5	6.5	6.7	7.0	6.9	6.3	6.2	7.0	6.8	7.0	6.7	6.7
Overall Ni Recovery	%	--	--	--	88.1	88.1	88.3	88.3	88.3	88.7	88.7	88.4	88.1	88.1	89.0	89.1	88.1	88.3	88.0	88.5	88.4
Ni Recovered	mlbs	--	--	--	80	105	111	105	105	113	106	91	91	131	157	162	143	150	157	161	148
Payable Ni	mlbs	--	--	--	75	99	105	99	99	107	100	86	86	123	148	153	135	141	148	152	140
Revenue	**US$M**	**--**	**--**	**--**	**659**	**866**	**918**	**868**	**869**	**932**	**879**	**751**	**755**	**1,079**	**1,297**	**1,341**	**1,181**	**1,237**	**1,293**	**1,333**	**1,223**
Operating Costs																					
Mining Costs	US$M	--	--	--	82	83	100	106	107	102	110	95	109	149	215	224	191	239	225	207	209
Processing Costs	US$M	--	--	--	136	148	148	146	146	145	143	139	142	210	212	211	216	218	224	220	216
G&A Costs	US$M	--	--	--	47	47	47	47	47	47	47	47	47	61	61	61	61	61	61	61	61
Concentrate Transport Costs	US$M	--	--	--	9	12	13	12	12	13	12	11	11	15	18	19	17	17	18	19	17
Total Operating Costs	**US$M**	**--**	**--**	**--**	**274**	**290**	**308**	**312**	**312**	**307**	**312**	**292**	**308**	**436**	**506**	**516**	**485**	**535**	**528**	**507**	**504**
Royalties																					
Total Revenue	US$M	--	--	--	659	866	918	868	869	932	879	751	755	1,079	1,297	1,341	1,181	1,237	1,293	1,333	1,223
Less: Transportation Costs	US$M	--	--	--	(9)	(12)	(13)	(12)	(12)	(13)	(12)	(11)	(11)	(15)	(18)	(19)	(17)	(17)	(18)	(19)	(17)
Total Net Revenue	US$M	--	--	--	649	853	905	856	856	919	867	740	745	1,064	1,278	1,322	1,164	1,220	1,275	1,314	1,206
NSR Royalty	%	--	--	--	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Royalties	**US$M**	**--**	**--**	**--**	**6**	**9**	**9**	**9**	**9**	**9**	**9**	**7**	**7**	**11**	**13**	**13**	**12**	**12**	**13**	**13**	**12**
Cash Costs																					
C1 Costs*	US$/lb Ni	--	--	--	3.93	3.26	3.26	3.46	3.46	3.21	3.42	3.70	3.86	3.83	3.71	3.67	3.88	4.06	3.86	3.63	3.89
All-in Sustaining Cost (AISC)**	US$/lb Ni	--	--	--	4.96	3.51	3.70	3.94	4.38	3.40	3.84	3.92	4.38	4.02	4.83	4.29	4.21	4.55	4.14	3.93	4.23
Capital Expenditures																					
Initial Capital	**US$M**	**436**	**654**	**1,091**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**
Sustaining Capital	**US$M**	**--**	**--**	**--**	**74**	**16**	**38**	**41**	**87**	**11**	**34**	**12**	**38**	**12**	**159**	**85**	**32**	**57**	**28**	**33**	**35**
BC Hydro Integration	**US$M**	**9**	**--**	**--**	**(1)**	**(1)**	**(1)**	**(1)**	**(1)**	**(1)**	**(1)**	**(1)**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**
Expansion Capital	**US$M**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**763**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**
Closure Costs	**US$M**	**--**	**--**	**--**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**

Table 22-3: Projected LOM Post-Tax Unlevered Free Cash Flow (Continued)

Dollar figures in Real 2023 US$M unless otherwise noted		Project Year																		
	Unit	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	Total/Avg.
Macro Assumptions																				
Nickel Price - Flat	US$/lb	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75	8.75
Exchange Rate	C$:US$	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76
Free Cash Flow Valuation																				
Revenue	**US$M**	**1,187**	**1,164**	**1,192**	**1,158**	**1,219**	**1,295**	**1,279**	**1,265**	**1,322**	**1,148**	**1,050**	**479**	**--**	**--**	**--**	**--**	**--**	**--**	**31,238**
Operating Costs	US$M	(499)	(495)	(479)	(482)	(488)	(487)	(478)	(437)	(452)	(460)	(433)	(206)	--	--	--	--	--	--	(12,130)
Royalties	US$M	(12)	(11)	(12)	(11)	(12)	(13)	(13)	(12)	(13)	(11)	(10)	(5)	--	--	--	--	--	--	(308)
EBITDA	**US$M**	**676**	**657**	**702**	**665**	**719**	**795**	**788**	**816**	**857**	**676**	**607**	**268**	**--**	**--**	**--**	**--**	**--**	**--**	**18,800**
Initial Capital	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	(2,182)
Sustaining Capital	US$M	(45)	(47)	(28)	(49)	(47)	(55)	(27)	(15)	(14)	(16)	(37)	(9)	--	--	--	--	--	--	(1,181)
BC Hydro Integration	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Expansion Capital	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	(763)
Closure Costs	US$M	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(19)	(5)	(5)	(74)	(73)	(33)	(1)	(1)	(3)	(284)
Changes in Working Capital	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Pre-Tax Unlevered Free Cash Flow	**US$M**	**628**	**607**	**670**	**613**	**669**	**738**	**759**	**798**	**840**	**641**	**565**	**255**	**(74)**	**(73)**	**(33)**	**(1)**	**(1)**	**(3)**	**14,390**
Pre-Tax Cumulative Unlevered Free Cash Flow	*US$M*	*7,422*	*8,028*	*8,699*	*9,312*	*9,981*	*10,718*	*11,477*	*12,275*	*13,115*	*13,756*	*14,321*	*14,576*	*14,501*	*14,428*	*14,395*	*14,394*	*14,392*	*14,390*	*14,390*
Tax Payable	**US$M**	**226**	**220**	**239**	**225**	**245**	**272**	**272**	**285**	**302**	**231**	**209**	**88**	**(26)**	**(24)**	**(22)**	**--**	**--**	**--**	**4,791**
Pre-Tax Unlevered Free Cash Flow	**US$M**	**402**	**387**	**431**	**388**	**424**	**466**	**486**	**513**	**538**	**409**	**355**	**166**	**(48)**	**(49)**	**(11)**	**(1)**	**(1)**	**(3)**	**9,599**
Pre-Tax Cumulative Unlevered Free Cash Flow	*US$M*	*5,148*	*5,535*	*5,966*	*6,355*	*6,779*	*7,245*	*7,731*	*8,243*	*8,781*	*9,191*	*9,546*	*9,713*	*9,664*	*9,615*	*9,604*	*9,603*	*9,601*	*9,599*	*9,599*
Production																				
Mill Feed	kt	58,089	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	59,130	43,148	23,042	--	--	--	--	--	--	1,487,721
Ni Grade	%	0.12	0.12	0.12	0.12	0.13	0.13	0.13	0.13	0.14	0.12	0.11	0.12	--	--	--	--	--	--	0.13
Ni Recovery - Awaruite Concentrate	%	82.6	82.8	83.2	83.2	83.7	83.6	83.4	83.7	83.6	81.3	80.8	81.4	--	--	--	--	--	--	82.2
Ni Recovery - MHP	%	6.3	6.2	6.1	6.1	5.8	5.9	6.0	5.8	5.9	6.9	7.1	6.9	--	--	--	--	--	--	6.5
Overall Ni Recovery	%	88.9	89.1	89.3	89.3	89.6	89.5	89.4	89.5	89.5	88.2	87.9	88.2	--	--	--	--	--	--	88.7
Ni Recovered	mlbs	144	141	144	140	147	157	155	153	160	139	127	58	--	--	--	--	--	--	3,781
Payable Ni	mlbs	136	133	136	132	139	148	146	145	151	131	120	55	--	--	--	--	--	--	3,570
Revenue	**US$M**	**1,187**	**1,164**	**1,192**	**1,158**	**1,219**	**1,295**	**1,279**	**1,265**	**1,322**	**1,148**	**1,050**	**479**	**--**	**--**	**--**	**--**	**--**	**--**	**31,238**
Operating Costs																				
Mining Costs	US$M	213	212	196	200	207	201	191	152	163	166	143	80	--	--	--	--	--	--	4,676
Processing Costs	US$M	208	206	205	204	203	207	208	206	209	216	214	93	--	--	--	--	--	--	5,398
G&A Costs	US$M	61	61	61	61	61	61	61	61	61	61	61	27	--	--	--	--	--	--	1,619
Concentrate Transport Costs	US$M	17	16	17	16	17	18	18	18	18	16	15	7	--	--	--	--	--	--	437
Total Operating Costs	**US$M**	**499**	**495**	**479**	**482**	**488**	**487**	**478**	**437**	**452**	**460**	**433**	**206**	**--**	**--**	**--**	**--**	**--**	**--**	**12,130**
Royalties																				
Total Revenue	US$M	1,187	1,164	1,192	1,158	1,219	1,295	1,279	1,265	1,322	1,148	1,050	479	--	--	--	--	--	--	31,238
Less: Transportation Costs	US$M	(17)	(16)	(17)	(16)	(17)	(18)	(18)	(18)	(18)	(16)	(15)	(7)	--	--	--	--	--	--	(437)
Total Net Revenue	US$M	1,170	1,148	1,176	1,142	1,202	1,277	1,261	1,247	1,303	1,131	1,036	472	--	--	--	--	--	--	30,801
NSR Royalty	%	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	--	--	--	--	--	--	1.0
Royalties	**US$M**	**12**	**11**	**12**	**11**	**12**	**13**	**13**	**12**	**13**	**11**	**10**	**5**	**--**	**--**	**--**	**--**	**--**	**--**	**308**
Cash Costs																				
C1 Costs*	US$/lb Ni	3.96	4.00	3.81	3.92	3.79	3.59	3.57	3.34	3.31	3.80	3.90	4.05	--	--	--	--	--	--	3.70
All-in Sustaining Cost (AISC)**	US$/lb Ni	4.38	4.44	4.10	4.37	4.21	4.04	3.85	3.53	3.49	4.14	4.31	7.56	--	--	--	--	--	--	4.17
Capital Expenditures																				
Initial Capital	**US$M**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**2,182**
Sustaining Capital	**US$M**	**45**	**47**	**28**	**49**	**47**	**55**	**27**	**15**	**14**	**16**	**37**	**9**	**--**	**--**	**--**	**--**	**--**	**--**	**1,181**
BC Hydro Integration	**US$M**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**
Expansion Capital	**US$M**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**763**
Closure Costs	**US$M**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**19**	**5**	**5**	**74**	**73**	**33**	**1**	**1**	**3**	**284**

* C1 costs consist of operating cash costs plus treatment and refining charges, and transportation cost.
** AISC consist of total cash costs plus royalties, sustaining capital, BC hydro integration, and closure cost.

22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax 8% NPV and IRR of the project, using the following variables: discount rate, nickel prices, metallurgical recovery, operating costs, and initial capital costs. Pre-tax sensitivity analysis results are summarized in Table 22-4, Figure 22-2, and Figure 22-3. Post-tax sensitivity analysis results are summarized in Table 22-5, Figure 22-4, and Figure 22-5. Analysis revealed that the project's NPV and IRR are most sensitive to changes in nickel price and metallurgical recovery, and due to the 29-year mine life, the project's NPV is more sensitive to operating cost than initial capital cost.

Table 22-4: Pre-Tax Sensitivity Analysis Results

Pre-Tax NPV (US$M) Sensitivity To Discount Rate						Pre-Tax IRR % Sensitivity To Discount Rate					
	Nickel Price						Nickel Price				
Discount Rate	(20%)	(10%)	--	10%	20%	Discount Rate	(20.0%)	(10.0%)	--	10.0%	20.0%
5%	2,531	3,896	5,262	6,627	7,993	5%	12.7%	16.1%	19.1%	22.0%	24.7%
8%	1,104	2,014	2,923	3,833	4,743	8%	12.7%	16.1%	19.1%	22.0%	24.7%
10%	520	1,235	1,950	2,665	3,380	10%	12.7%	16.1%	19.1%	22.0%	24.7%

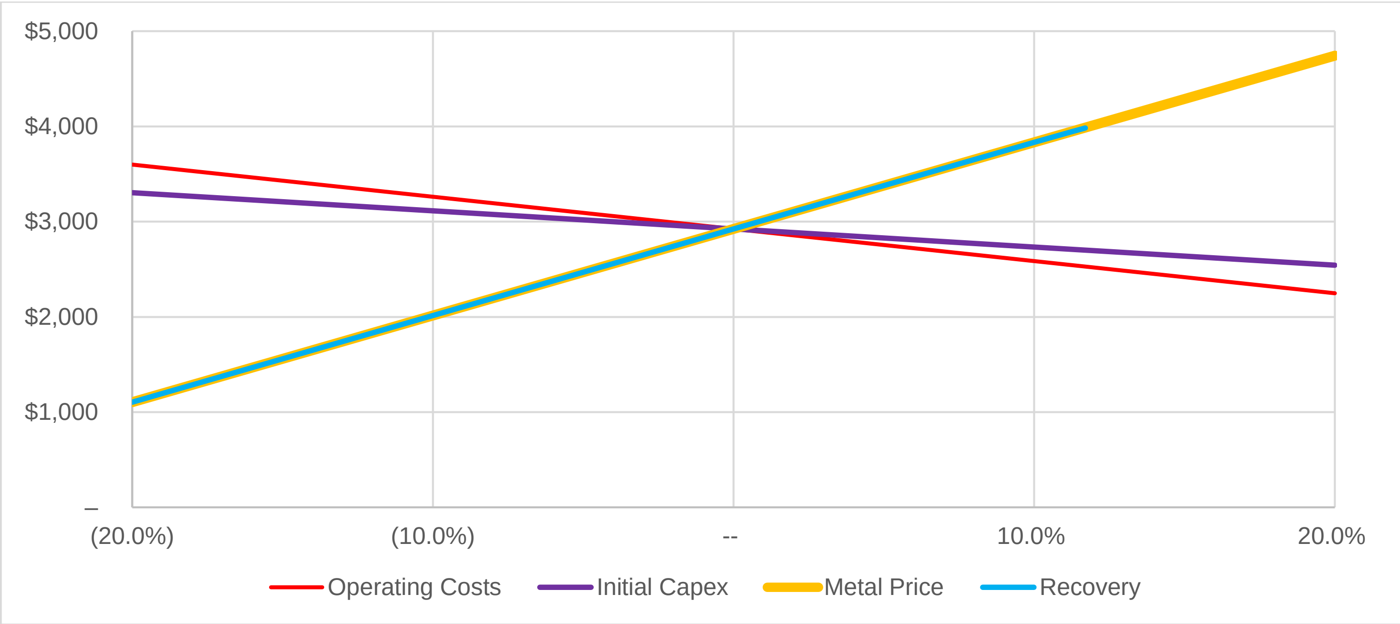
Pre-Tax NPV (US$M) Sensitivity To Initial Capital Cost						Pre-Tax IRR Sensitivity To Initial Capital Cost					
	Nickel Price						Nickel Price				
Initial Capital	(20%)	(10%)	--	10%	20%	Initial Capital	(20.0%)	(10.0%)	--	10.0%	20.0%
(20%)	1,485	2,394	3,304	4,213	5,123	(20%)	15.5%	19.4%	23.0%	26.3%	29.5%
(10%)	1,295	2,204	3,114	4,023	4,933	(10%)	14.0%	17.6%	20.9%	24.0%	26.9%
--	1,104	2,014	2,923	3,833	4,743	--	12.7%	16.1%	19.1%	22.0%	24.7%
10%	914	1,824	2,733	3,643	4,552	10%	11.6%	14.8%	17.7%	20.4%	22.9%
20%	724	1,633	2,543	3,452	4,362	20%	10.7%	13.7%	16.4%	19.0%	21.4%

Pre-Tax NPV (US$M) Sensitivity To Operating Cost						Pre-Tax IRR Sensitivity To Operating Cost					
	Nickel Price						Nickel Price				
Operating Cost	(20%)	(10%)	--	10%	20%	Operating Cost	(20.0%)	(10.0%)	--	10.0%	20.0%
(20%)	1,780	2,689	3,599	4,508	5,418	(20%)	15.2%	18.3%	21.2%	24.0%	26.6%
(10%)	1,442	2,352	3,261	4,171	5,080	(10%)	14.0%	17.2%	20.2%	23.0%	25.7%
--	1,104	2,014	2,923	3,833	4,743	--	12.7%	16.1%	19.1%	22.0%	24.7%
10%	767	1,676	2,586	3,495	4,405	10%	11.4%	14.9%	18.0%	21.0%	23.8%
20%	429	1,338	2,248	3,157	4,067	20%	10.0%	13.6%	16.9%	20.0%	22.8%

Pre-Tax NPV (US$M) Sensitivity To Recovery						Pre-Tax IRR Sensitivity To Recovery					
	Nickel Price						Nickel Price				
Recovery	(20%)	(10%)	--	10%	20%	Recovery	(20.0%)	(10.0%)	--	10.0%	20.0%
(20%)	(351)	377	1,104	1,832	2,560	(20%)	6.3%	9.7%	12.7%	15.4%	17.9%
(10%)	377	1,195	2,014	2,832	3,651	(10%)	9.7%	13.1%	16.1%	18.8%	21.4%
--	1,104	2,014	2,923	3,833	4,743	--	12.7%	16.1%	19.1%	22.0%	24.7%
10%	1,832	2,832	3,833	4,833	5,834	10%	15.4%	18.8%	22.0%	25.0%	27.8%
20%	N/A	N/A	N/A	N/A	N/A	20%	N/A	N/A	N/A	N/A	N/A

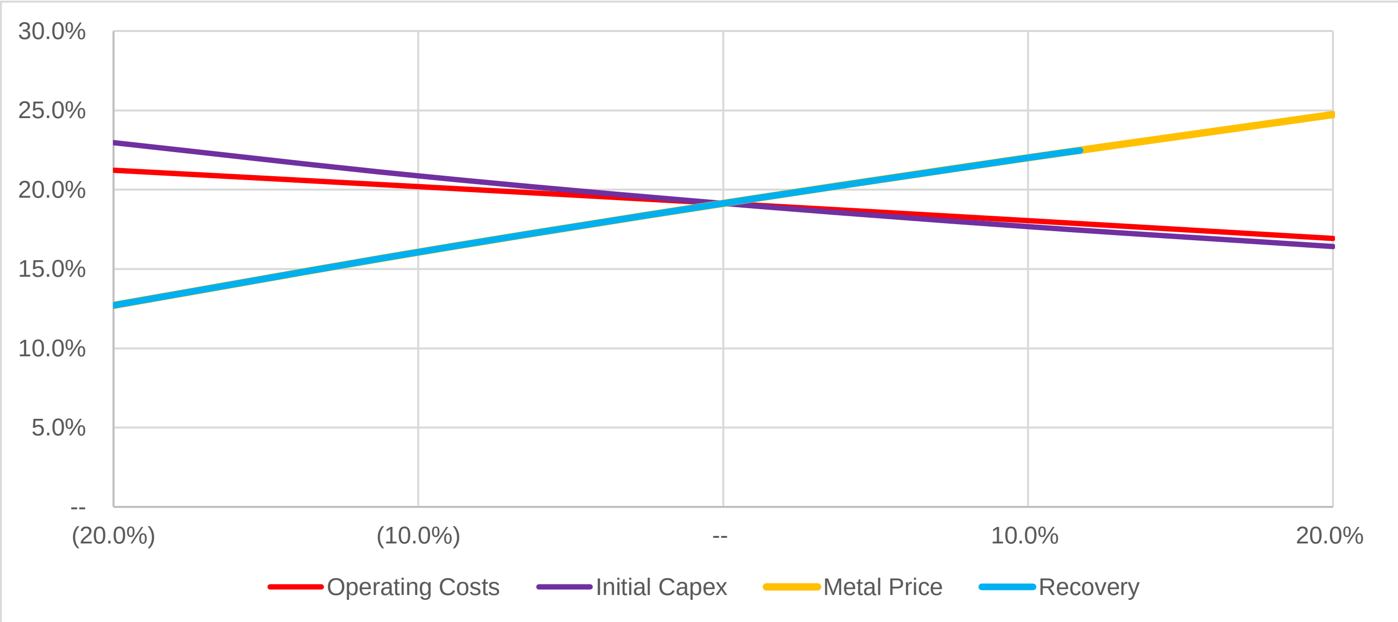
Source: Ausenco, 2023.

Figure 22-2: Pre-Tax NPV 8% Sensitivity Chart (US$M)



Source: Ausenco, 2023.

Figure 22-3: Pre-Tax IRR Sensitivity Chart



Source: Ausenco, 2023.
Notes:
- In Figure 22-2 and Figure 22-3 chart lines for metal price and metallurgical recovery overlap.
- In Figure 22-2 and Figure 22-3 chart lines for metallurgical recovery end as recovery reaches 100%.

Table 22-5: Post-Tax Sensitivity Analysis Results

Post-Tax NPV (US$M) Sensitivity To Discount Rate

Discount Rate	Nickel Price				
	(20%)	(10%)	0%	10%	20%
5%	1,840	2,714	3,583	4,453	5,320
8%	837	1,427	2,010	2,593	3,173
10%	415	883	1,345	1,805	2,262

Post-Tax IRR Sensitivity To Discount Rate

Discount Rate	Nickel Price				
	(20%)	(10%)	0%	10%	20%
5%	13.0%	16.0%	18.6%	21.1%	23.4%
8%	13.0%	16.0%	18.6%	21.1%	23.4%
10%	13.0%	16.0%	18.6%	21.1%	23.4%

Post-Tax NPV (US$M) Sensitivity To Initial Capital Cost

Initial Capital	Nickel Price				
	(20%)	(10%)	0%	10%	20%
(20%)	1,051	1,635	2,217	2,796	3,375
(10%)	945	1,531	2,114	2,695	3,274
--	837	1,427	2,010	2,593	3,173
10%	728	1,321	1,907	2,490	3,071
20%	618	1,214	1,803	2,386	2,969

Post-Tax IRR Sensitivity To Initial Capital Cost

Initial Capital	Nickel Price				
	(20%)	(10%)	0%	10%	20%
(20%)	15.4%	18.8%	21.8%	24.6%	27.2%
(10%)	14.1%	17.2%	20.1%	22.7%	25.2%
--	13.0%	16.0%	18.6%	21.1%	23.4%
10%	12.0%	14.9%	17.4%	19.7%	21.9%
20%	11.2%	13.9%	16.3%	18.6%	20.6%

Post-Tax NPV (US$M) Sensitivity To Operating Cost

Operating Cost	Nickel Price				
	(20%)	(10%)	0%	10%	20%
(20%)	1,276	1,861	2,444	3,024	3,604
(10%)	1,057	1,644	2,227	2,809	3,388
--	837	1,427	2,010	2,593	3,173
10%	615	1,209	1,794	2,376	2,957
20%	392	989	1,577	2,160	2,742

Post-Tax IRR Sensitivity To Operating Cost

Operating Cost	Nickel Price				
	(20%)	(10%)	0%	10%	20%
(20%)	15.2%	17.9%	20.4%	22.8%	25.0%
(10%)	14.1%	17.0%	19.6%	22.0%	24.2%
--	13.0%	16.0%	18.6%	21.1%	23.4%
10%	11.7%	14.9%	17.7%	20.2%	22.6%
20%	10.5%	13.8%	16.7%	19.3%	21.8%

Post-Tax NPV (US$M) Sensitivity To Recovery

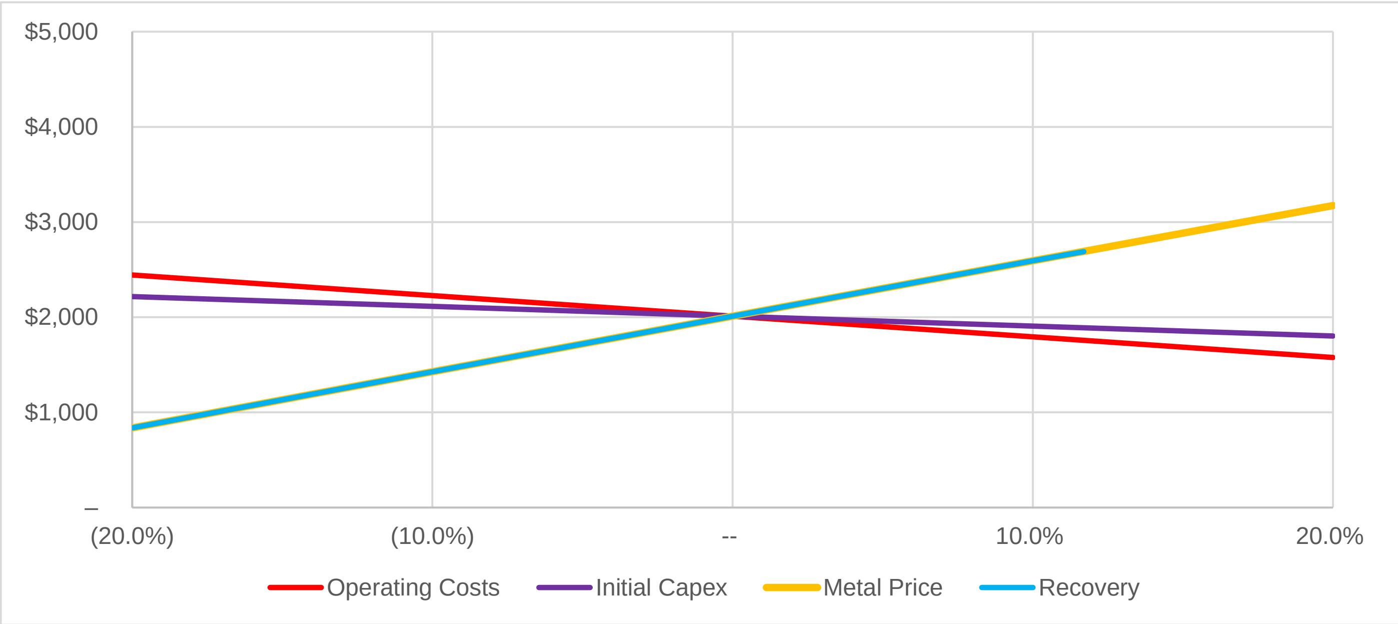
Recovery	Nickel Price				
	(20%)	(10%)	0%	10%	20%
(20%)	(135)	357	837	1,310	1,777
(10%)	357	896	1,427	1,952	2,476
--	837	1,427	2,010	2,593	3,173
10%	1,310	1,952	2,593	3,231	3,868
20%	N/A	N/A	N/A	N/A	N/A

Post-Tax IRR Sensitivity To Recovery

Recovery	Nickel Price				
	(20%)	(10%)	0%	10%	20%
(20%)	7.1%	10.2%	13.0%	15.4%	17.6%
(10%)	10.2%	13.3%	16.0%	18.4%	20.6%
--	13.0%	16.0%	18.6%	21.1%	23.4%
10%	15.4%	18.4%	21.1%	23.6%	26.0%
20%	N/A	N/A	N/A	N/A	N/A

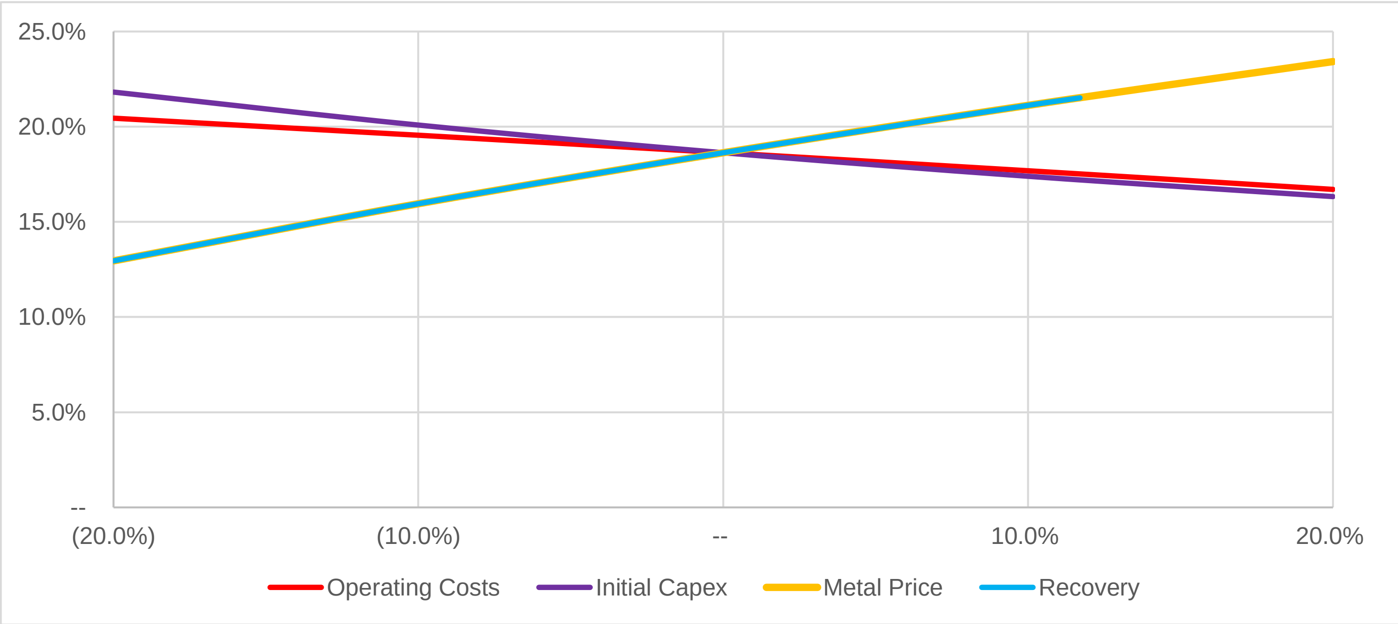
Source: Ausenco, 2023.

Figure 22-4: Post-Tax NPV 8% Sensitivity Chart (US$M)



Source: Ausenco, 2023.

Figure 22-5: Post-Tax IRR Sensitivity Chart



Source: Ausenco, 2023.
Notes:
- In Figure 22-4 and Figure 22-5 chart lines for metal price and metallurgical recovery overlap.
- In Figure 22-4 and Figure 22-5 chart lines for metallurgical recovery end as recovery reaches 100%.

23 ADJACENT PROPERTIES

Mineral properties that lie adjacent to the Decar Property are held by Mr. Brynensen and Mr. Bell, Grid Battery Metals Inc. (Grid Battery), Mr. Kelly Brent Funk (MAC Property), Mr. Richard Young, and Mr. Aimin Liao. The location of these claim blocks relative to the Decar Property are shown in Figure 23-1 and three of these properties are described in more detail below. No information was found for the claims owned by Mr. Young and Mr. Liao.

The Mount Sidney Williams and MAC properties are located adjacent to and west of the Decar Property, though only the Mount Sidney Williams property has documented occurrences of awaruite that may be like the mineralization found at Decar. The MAC property, on the other hand, hosts a molybdenum-copper (Mo-Cu) porphyry deposit.

Information for the properties discussed below has not been verified by any of the QPs on this report and is not necessarily indicative of mineralization on the Decar Property.

23.1 Brynelsen and Bell Claims

Mr. Dal Stuart Brynelsen and Mr. John Malcolm Bell hold the Mount Sidney Williams property, which is partially enclosed by the western side of the Decar Property and also borders Grid Battery claims to the west. The property is 50% owned by Mr. Brynelsen and 50% by Mr. Bell.

The Mount Sidney Williams property is underlain by Cache Creek terrane rocks that includes Trembleur ultramafite, the same unit that hosts awaruite mineralization on the Decar Property. Prospecting work has defined a 500 m by 1500 m zone of serpentinized ultramafic rock with fine-grained, disseminated, awaruite in addition to localized occurrences of coarser-grained awaruite that is visible to the naked eye (Mowat, 2013).

Information in the assessment report by Mowat (2013) has not been validated by any of the QPs for this technical report and is not necessarily indicative of mineralization on the Decar Property.

23.2 MAC Property

The MAC property lies immediately west of the Decar Property, is owned by Mr. Kelly Funk, and is currently under option to Transforma Resources Corp (Hanson, 2023). The property hosts a Mo-Cu porphyry deposit as well as ultramafic rocks that are prospective for awaruite-style mineralization.

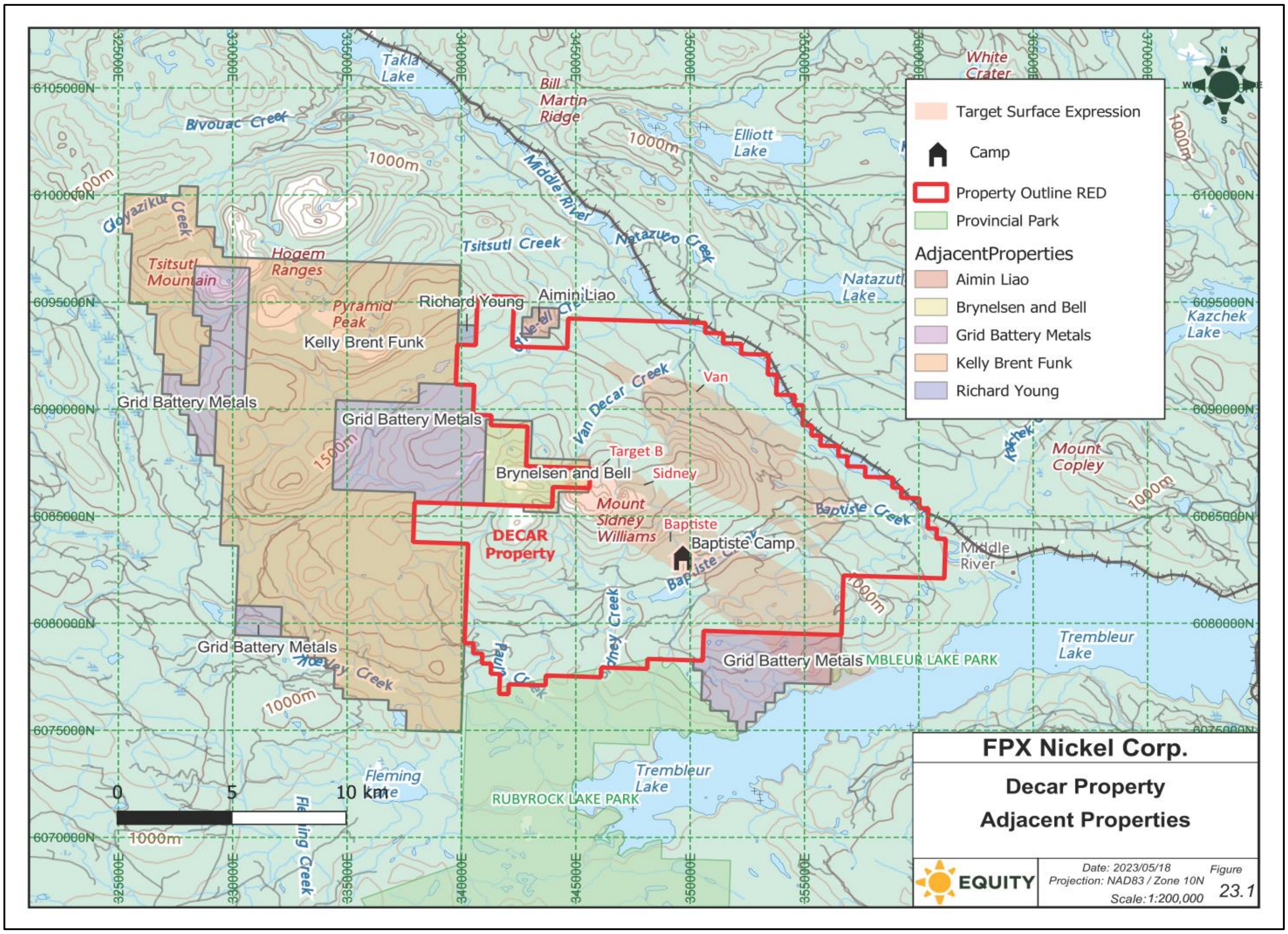
The Mo-Cu porphyry deposit was drilled off between 1995 and 2010, culminating in a maiden resource estimate (Giroux and Moore, 2012) that does not comply with NI 43-101 reporting standards (Hanson, 2023). No further work has been done on this porphyry deposit and this mineralization is not considered material to the Decar Property.

More recent work on the MAC property has focused on the potential to host awaruite deposits similar to those found on the Decar Property (Hanson, 2023). Rock sampling in 2012 and 2021 reported the presence of visible awaruite and included the collection of serpentinized ultramafic rocks with similar whole rock nickel and cobalt contents to mineralized rocks on the Decar Property. Analysis of magnetic separates generated from these ultramafic rocks returned between <0.001% DTR Ni to 0.076%, the latter suggesting the presence of awaruite. More work is required to determine the host mineral for these elevated DTR nickel contents (Hanson, 2023).

23.3 Grid Battery Claims

Grid Battery owns claims immediately to the south and west of the Decar Property, in addition to claims blocks located to the west of the MAC property. All these claims cover parts of the Cache Creek terrane that are prospective for awaruite mineralization like that found on the Decar Property. Grid Battery completed prospecting and soil sampling programs in 2021 but did not provide a summary of results and has completed no work since then.

Figure 23-1: Location of Adjacent Properties



Source: Equity Exploration, 2023.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

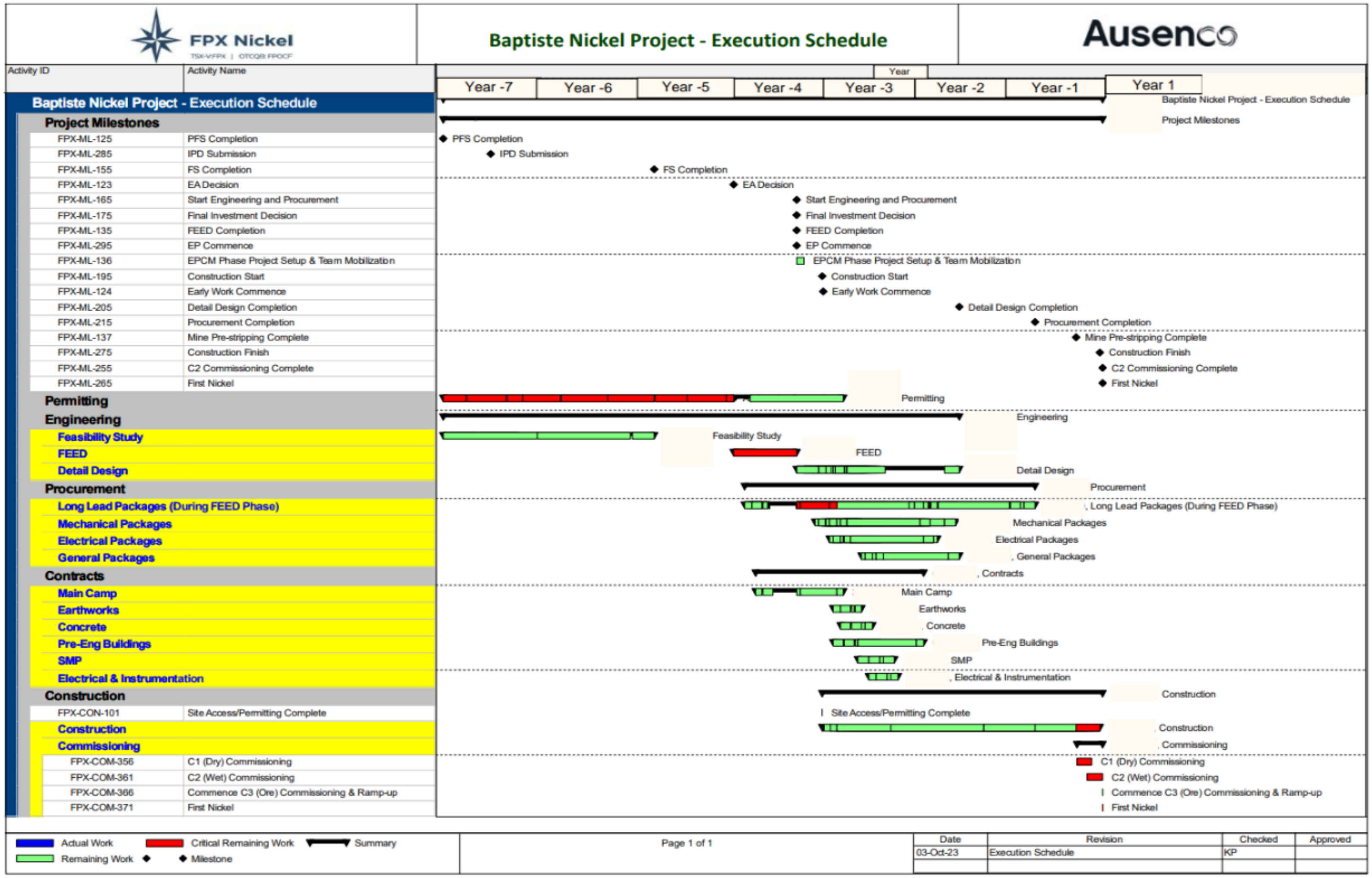
The Project Execution Plan (PEP) is a governing document that establishes the means to execute, monitor, and control the execution phase of the Baptiste Nickel Project. The PEP will serve as the main communication tool to ensure the project team is aware and knowledgeable of the Project objectives and how they will be accomplished. The Project is intended to be constructed in two distinct phases, an initial installation (Phase 1) which includes a mine, concentrator, and support facilities at the Baptiste site and an expansion (Phase 2) that includes an increase to mine and process plant capacity. The PEP does not include the execution planning for the off-site hydrometallurgical refinery (Refinery Option) described in Section 24.2.

24.1.1 Execution Strategy

An EPCM contracting strategy will be utilized to deliver the detailed engineering and execution phases of the Project. An EPCM contract led by a FPX-selected Contractor will generally include the processing facilities and on-site infrastructure, as well as integration of with FPX's third-party specialist consultants for waste and water management, off-site roads, off-site power, and mining.

Following the PFS, the FS and EA/permitting work streams will be advanced in tandem, with the EA/permitting work stream representing the project's critical path. Following completion of the FS, front end engineering and design (FEED) will be conducted to suitably advance engineering to support formal permitting efforts, as well as advance the project's business case to support a final investment decision (FID). Following the FID, the project will immediately commence detailed engineering, followed shortly thereafter by early works construction activities at the start of Year -3.

Figure 24-1: Summary Execution Schedule



Source: Ausenco, 2023.

24.1.2 Project Risk Identification

Risk management is an ongoing exercise that will be performed regularly throughout the life of the Baptiste Nickel Project. During the PFS a Risk-Hazard Identification Workshop was held and a summary document of the identified risks was created. The FS and detailed engineering phase will prioritize mitigation, and elimination where possible, of the operational risks identified.

24.1.3 Project Team

The project team is organized based on an integrated team approach, minimizing the duplication of roles and activities between the Owner's Team and their major delivery partners. A proposed project organization will be developed during the FS.

FPX is performing or managing a considerable part of the project scope, engaging with several delivery contractors to execute multiple distinct construction scopes of work. Some of these scopes include the mine design, waste and water management, off-site access roads, off-site power transmission line, and mining.

The lead consultant and FPX will work in tandem to ensure that all delivery partners are familiar with the work, and key persons will be established for each project participant to ensure efficient coordination.

24.1.4 Project Alignment

The project alignment strategy aims to create shared understanding of the Project vision and strategy to enable FPX's internal and external stakeholders to achieve the Project objectives. The project delivery team operates as one team with defined responsibilities, accountabilities, and authorities. The team is established and supported to deliver "Best for Project" outcomes in line with FPX's expectations and critical success factors.

Establishment of the delivery team working relationships and agreeing acceptable desired outcomes will be done in facilitated alignment sessions.

The alignment effort will be concentrated at the front-end of the project, although ongoing activities are planned throughout to increase overall effectiveness, commitment, and cohesiveness of project team members.

24.1.5 Sponsor Group

A sponsor group will be formed to reinforce corporate commitment to the Project as it passes through its various phases.

Key activities include:

- Directing the business objectives for the participants to achieve "Best for Project" outcomes;
- Providing corporate commitment to achieving the desired outcomes for the Project;
- Reinforcing common purpose in achieving the Project goals;
- Managing third-party events outside of the control of the project team;

- Providing corporate recognition and reward for performance; and
- Supporting the Project in resolution of issues.

The sponsor group will comprise senior executives from the lead consultant and FPX. The sponsor group will stay abreast of events and issues on and around the Project. The principal responsibility of each member in their role on the sponsor group is directed at ensuring that the Project is guided, supported, and encouraged to achieve the project objectives. Each member's association with their own organization is secondary to their responsibility to support the Project.

The sponsor group will meet monthly.

24.1.6 Governance

The project delegated authority matrix (DAM) including agreed levels of signing authorities for the Project will be developed during the FEED phase.

24.1.7 External Relations

All external communications and engagements will be handled by FPX's Project Director, or designated FPX team member. No other member of the project is authorized to engage with the local community, government, media, or any other entity on behalf of the Project.

24.1.8 Construction Execution Strategy

24.1.8.1 Construction Sequence

After the FS stage is completed, there will be a period of FEED and permitting that will be completed prior to a final investment decision (FID) which will trigger the commencement of detailed engineering, as well as approve the first mobilization to site. These FEED and permitting activities include the following main tasks:

- Environmental assessment and permitting;
- Advancement of critical path engineering activities;
- Procurement of long lead items (mining fleet, Ball/SAG mills, etc.); and
- Award of key construction contracts (Camp Construction, Site Civil Works).

These project development activities will all be completed prior to first mobilization to site. Timing of mobilization is contingent on FPX receiving Provincial and Federal environmental assessment decisions and required permit authorizations, as described in Sections 20.2.1 and 20.2.2. No sitework be allowed to occur until the required permits are in hand. This includes early mobilization and staging equipment on site, early site preparations or stockpiling of construction materials.

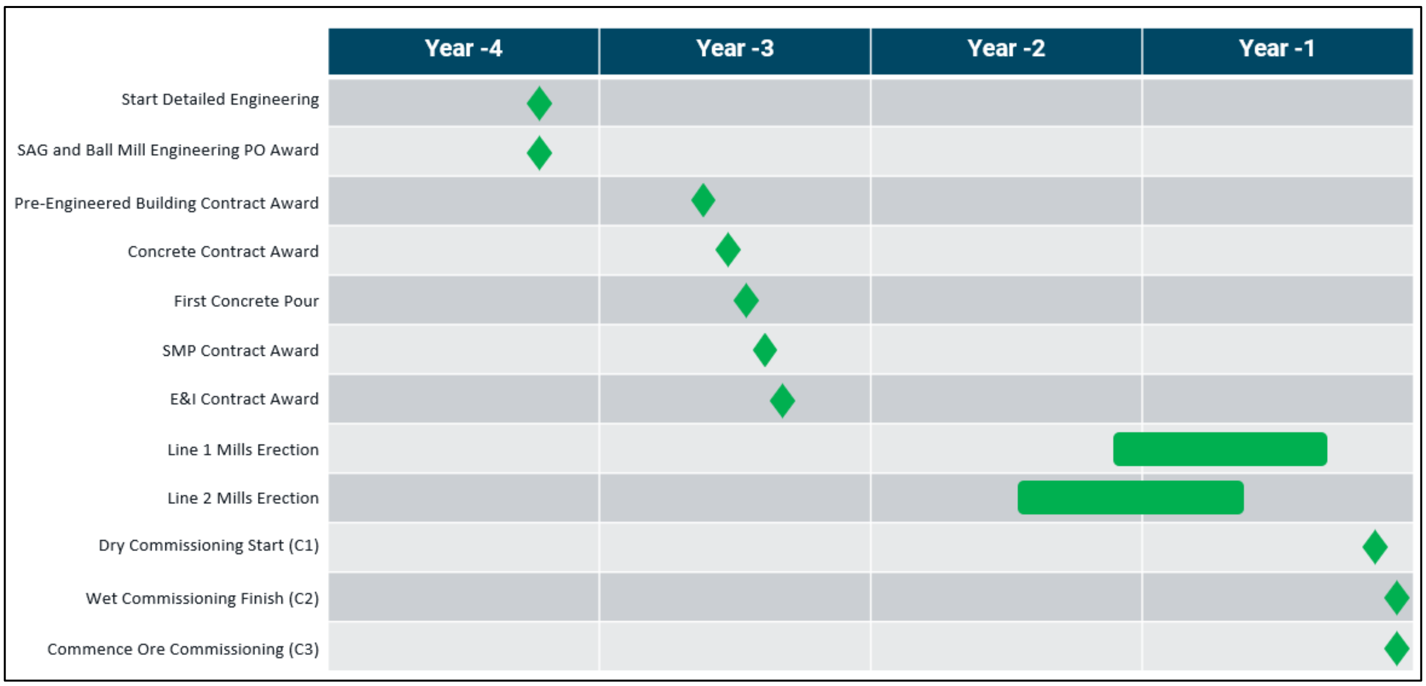
Once the permits are received, the first contractors to mobilize will be the camp construction contractors and the early civil works contractors responsible for the clearing and grubbing of the specific site works boundaries.

As the clearing and grubbing activities continue, heavy civil work will follow behind to strip the topsoil and organics and stockpile in designated areas for future remediation works. Temporary water management catchments and ditches will also be developed as the civil works continue in the pit, the tailings facility dam footprints, as well as the process plant pad development.

After the early civil works are completed there will be three main work-fronts on the project property: The mining works will continue the pit development, generating, and stockpiling waste rock material that will be used for the construction of the tailings facility dam. The tailings facility earthworks will include foundation preparation, construction of downstream water management ponds, and placing and compacting hauled overburden and waste rock to construct the dams. The process plant works will begin with concrete works in the summer of Year -3 for building/major equipment foundations and construction will be continuous until commissioning activities begin in September Year -1 prior to first concentrate in December Year -1.

Figure 24-2 lists key Lead Consultant EPCM milestone dates.

Figure 24-2: Lead Consultant EPCM Milestone Dates



Source: Ausenco, 2023

24.1.8.2 Site Laydown Requirements

An early priority for site construction will be the assembly of temporary and permanent storage warehouse facilities with sufficient space to store any goods with indoor storage requirements. Any goods or equipment that can be stored outdoors may be placed in an on-site outdoor lay down area, ideally located near the storage warehouse. The outdoor lay down area will be on level ground, with appropriate storage for site conditions.

A storage warehouse will be required for all materials requiring protection from the elements. An industrial building that is constructed early and is not immediately required for other purposes – such as the reagents building - may be used temporarily as a storage warehouse.

Both the site lay down and storage warehouse will need to obtain the necessary authorizations for the storage of any hazardous materials. The required security, protective and handling equipment will be on hand to allow for the temporary storage of hazardous materials whenever necessary.

24.1.8.3 Workforce Assumptions

The work schedule for the construction contractors' supervision and the EPCM contractor's personnel is based on 84 hours/working week, consisting of 7 days per week and 12 hours per day. This work schedule aligns with recent contractor's information received for similar projects in central and northern BC.

A labour loading forecast was developed for the construction phase, utilizing direct labour hours developed in the capital cost estimate. These direct hours were augmented with further direct hours back calculated from growth allowances and subcontracted scopes, and all were augmented with a factor for indirect hours. The total hours estimated were then spread over the execution duration to define the bed camp size at 1,569 beds. See Section 18.5.1 for a breakdown of the accommodation provision.

24.1.8.4 Shared Site Services

Services were identified during the prefeasibility study that were common across the work fronts during construction. It may be advantageous to offer these common services to the contractors both from a cost perspective, as well as to potentially award these site service contracts to local businesses. These services include:

- diesel fuel supply;
- temporary power supply;
- road maintenance/snow clearing;
- garbage removal;
- bussing workforce to/from the camp each day; and
- heavy construction equip

24.2 Refinery Option

24.2.1 Introduction

The Refinery Option is a conceptual study demonstrating Baptiste's strategic flexibility to produce nickel and cobalt for the EV battery material supply chain. This off-site refinery was developed to be discrete from the Base Case, demonstrating the economics of processes the awaruite concentrate and MHP products produced at the Baptiste Nickel Project. This section discusses the option of constructing and operating the refinery in central BC. Inclusion of this off-site refinery would not require any significant change to the design of the Baptiste mine, mine production schedule or main process plant. With the addition of this scope, the majority of the awaruite concentrate and all MHP concentrate will be sent to the refinery for further processing, with the remaining awaruite concentrate continuing to be marketed directly to stainless steel producers.

The refinery process consists of atmospheric leaching, pressure oxidation (POX) leaching, solvent extraction (SX), and crystallization and will produce three sellable products, including battery-grade nickel sulphate, cobalt hydroxide, and a copper cementation product (copper cement). The nickel sulphate, cobalt hydroxide, and copper cement will be transported initially by rail to buyers . A fourth product, sodium sulphate, is assumed to be free issued locally to kraft pulp mills for this study; however, may present an opportunity to add value in the future.

24.2.2 Property Description and Location

For the purposes of the Refinery option, the refinery is assumed to be in central BC, but separate from the mine and concentrator facilities at the Baptiste site. The exact location of the refinery has not been confirmed at the time of this report, but is assumed in a semi-urban setting in central BC, such as near the city of Prince George, which is located approximately 700 km NNE of Vancouver.

Prince George has a typical elevation of 575-650 masl in the urban area and hillier on the outskirts of the city. The coldest weather experienced in Prince George between 2009 and 2023 was -44°C and the warmest day was 38°C. On average, January sees the highest amount of snowfall with June having the greatest amount of rainfall.

Power, water, and labour is readily available at Prince George, a city that is a well-known and established hub for British Columbia’s resource industry.

24.2.3 Mineral Processing and Metallurgical Testing

24.2.3.1 Introduction

The Baptiste Nickel Project produces a high-grade awaruite concentrate through sequential magnetic separation and flotation operations. This awaruite concentrate consists primarily of the mineral awaruite, a nickel-iron alloy with nominal stoichiometry of Ni_3Fe. The high-nickel and low-impurity grades of the concentrate makes the awaruite concentrate a suitable candidate for further refining to produce battery grade nickel sulphate crystals for use in electric vehicle battery applications. As there are no existing industrial-scale operations that process awaruite-dominant nickel mineralization, there are no industrial-scale refineries processing awaruite-rich concentrates.

The main objectives of the metallurgical program evaluating the Refinery Option were:

- Develop a sulphuric acid leach system that simultaneously achieves high nickel recoveries (greater than 98%) and produces a pregnant leach solution (PLS) with low free acid and iron concentrations to reduce downstream purification costs. Given the high iron grade of the awaruite concentrate (24% Fe) relative to other high nickel grade intermediates, this requires effective iron management to re-precipitate the dissolved iron using elevated temperature conditions.
- Purify the PLS using established and industrially-proven technologies.

- Produce battery-grade nickel sulphate crystals from awaruite concentrate, using industry-standard unit operations, to demonstrate the technical viability of an awaruite-rich concentrate as feedstock into the EV battery material supply chain.

Metallurgical testing campaigns to refine the awaruite concentrate are a summarized in Table 24-1.

Table 24-1: Refining Metallurgical Testwork Summary

Year	Laboratory/Location	Major Testwork performed	Metallurgical Sample	Sample Size
2020	Sherritt Technologies, Fort Saskatchewan	Pressure leach	Flotation concentrates from 2020 ALS PEA test program	130 g
2023	Sherritt Technologies, Fort Saskatchewan	Pressure and atmospheric leach, solution purification, nickel sulphate crystallization	Flotation concentrates from 2022 Corem PFS pilot plant	6 kg

24.2.3.2 Sample Selection

The composition of the awaruite concentrate used in the two refining test programs is presented in Table 24-2. The 2020 and 2023 programs both used feedstocks of awaruite concentrates generated from flotation testwork as part of the PEA and PFS testwork programs, respectively. For comparison, the grades of 60% and 65% nickel flotation concentrates generated from the PFS flotation program are also presented. Notable items include:

- Two concentrate nickel grades, 60% and 65%, were evaluated in the PFS flotation program, with only a minor difference in recovery (0.3%) between the two. As such, the 65% Ni concentrate was considered for the Refinery Option design basis due to the materially lower magnesium concentration, which outweighs the small recovery loss.
- The 2020 feedstock has elevated copper levels due to the use of copper sulphate as an activator in flotation. Copper sulphate was proven not required in PFS flotation program.
- As discussed in Section 13.6.3.3.2, the bulk flotation concentrate from the 2022 Corem PFS pilot plant was generated by processing the significant nickel inventory held up in the regrind mill. As such, the 2023 program sample was expected to be a coarser fraction of awaruite. However, the 80% passing size of this material is comparable to the flotation concentrate generated from the 2021 LOM composite sample where hold-up in the regrind mill did not occur.
- The 80% passing size of the flotation concentrate is materially coarser than that of the upstream regrind size (~18 µm). This is due to the malleability of awaruite, which tends to deform to form flakes. Correspondingly, from a leaching kinetics perspective, the awaruite particles have a higher surface area to volume ratio than spheres of a similar diameter.

Table 24-2: List of Samples Used in Refining Metallurgical Testwork Campaigns

Sample	Ni (%)	Fe (%)	Co (%)	Mg (%)	Cu (%)	S (%)	Si (%)	Cr (%)	Mn (%)	Zn (%)	Al (%)	Ca (%)	P_{80} (µm)
2020 Refining Program	59.0	23.8	0.90	0.62	0.75	0.70	0.51	0.22	0.03	-	0.01	-	-
2023 Refining Program	66.3	24.6	1.08	0.56	0.43	0.44	0.49	0.21	0.02	<0.03	<0.04	-	160
PFS Flotation: 65%Ni	65.0	24.2	1.12	1.00	0.53	0.92	1.29	0.04	0.01	<0.01	0.03	0.01	-
PFS Flotation: 60%Ni	60.0	24.6	1.08	1.71	0.49	0.98	2.21	0.06	0.01	<0.01	0.05	0.01	159

24.2.3.3 Summary of Metallurgical Test Results

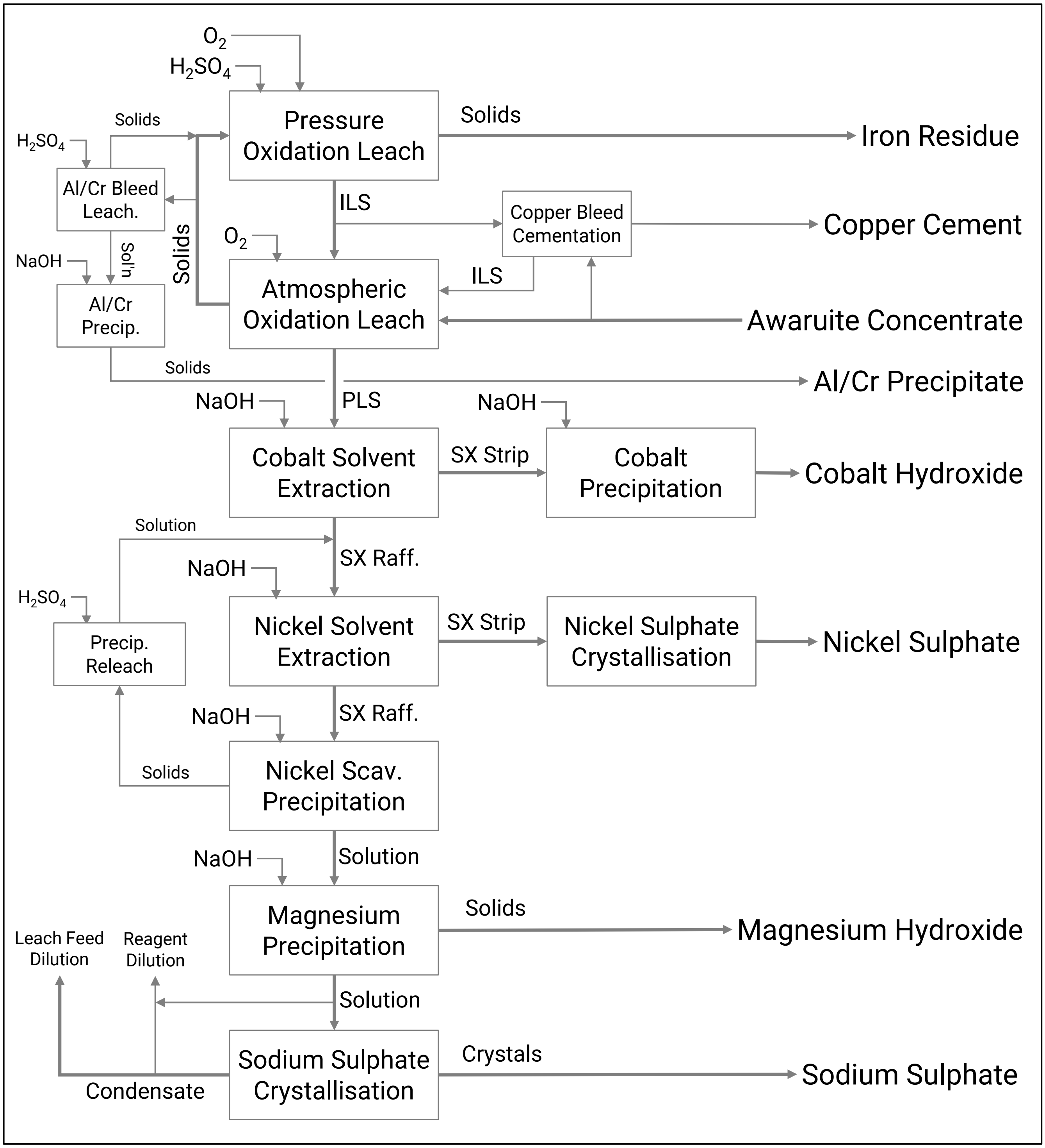
Counter-current leach conditions, consisting of a POX leach and an atmospheric oxidative leach, have been optimized in batch testing to yield an overall nickel leach recovery of greater than 99% while producing a low impurity PLS. Awaruite concentrate was demonstrated to be a sufficiently strong neutralization agent to obtain complete removal of free acid and iron in the atmospheric leach, permitting direct feeding of PLS to downstream SX operations. Awaruite concentrate also served as a strong reducing agent to produce a high-grade copper cement from the intermediate leach solution (ILS) as a byproduct.

Cobalt solvent extraction (CoSX) and nickel solvent extraction (NiSX) testing was limited to batch operation and provided a proof-of-concept for a purification scheme to produce nickel sulphate. CoSX was able to achieve sufficient Co over Ni selectively and NiSX was able to achieve sufficient Ni over Mg selectively to produce nickel sulphate crystals sufficiently low in these impurities. Other impurities in the PLS, though present only in trace quantities, include copper, zinc, manganese, and calcium, and were effectively managed in the purification scheme.

Nickel sulphate crystals were produced at purity levels meeting or exceeding current battery-grade specifications. This indicates that the awaruite concentrate is a suitable nickel intermediate for refining nickel sulphate to feed the battery electric vehicle industry.

The proposed flowsheet for refining the awaruite concentrate to nickel sulphate is shown in Figure 24-3, and is based on the metallurgical results presented in this section.

Figure 24-3: Block Flow Diagram of the Refinery Option Processing Route



Source: FPX, 2023.

24.2.3.4 Mineralogy

Dedicated mineralogy work was not completed on the awaruite concentrate. Inferred mineralogy based on the concentrate elemental grades (from the 2023 program) and mineralogy completed on the whole ore and flotation tailings (see Section 13) has been calculated to provide an indication of mineral concentrations and is presented in Table 24-3. The concentrate is dominated by awaruite with lesser concentrations of magnetite and serpentine. All sulphur is suspected to be present as nickel sulphides, predominantly pentlandite, though a lesser amount of heazlewoodite is expected as well. The unidentified chromium spinel has been assigned chromite stoichiometry for calculation purposes. Copper and cobalt are not shown but are associated with awaruite.

Table 24-3: Inferred mineralogy of Awaruite Concentrate

Mineral	Chemical Composition	Normalized Concentration (%)
Awaruite	Ni_3Fe	88.6
Magnetite	Fe_3O_4	7.1
Serpentine	$(Mg,Fe)_3Si_2O_5(OH)_4$	2.4
Pentlandite	$Fe_{4.5}Ni_{4.5}S_8$	1.4
Unidentified Cr-Spinel	$FeCr_2O_4$	0.5

24.2.3.5 Pressure Oxidation Leach

A total of twelve batch POX leach tests were completed in the 2023 refinery program. POX leaching of the awaruite concentrate at 150°C with an oxygen partial pressure of 375 kPa yielded nickel and cobalt extractions greater than 99% within 30 minutes, while effectively managing iron deportment (<0.1% Fe extraction) and terminal acid concentration (20 to 25 g/L H_2SO_4).

Typical leach results, with low and high discharge free acidities and the impact of recycled atmospheric leach residue are presented in Table 24-4. Elevated free acidities slightly increase nickel recovery, and this is suspected to be related to dissolution of the minor amount of nickel present as a nickel sulphide. The higher acidity conditions were selected as optimal due to the higher recoveries and limited consideration was given to the higher acidity as the downstream atmospheric oxidation leach could effectively manage this acidity (see Section 24.1.3.6). Similarly high nickel and cobalt extractions, and effective iron management, were observed when partially leached atmospheric leach residues were recycled to POX leach in combination with fresh awaruite concentrate, confirming the amenability of the awaruite concentrate to a two-stage, counter-current leach.

Table 24-4: Pressure Oxidation Leach Results Summary (2023 Program)

Test ID	Atm Residue (%feed w/w)	Leach Extraction (%)				Produced ILS (g/L)					
		Ni	Co	Cu	Mg	Ni	Fe	H_2SO_4	Cu	Co	Mg
1	0	98.8	98.0	86.0	94.6	67.7	0.08	9.2	0.41	1.14	0.56
2	0	99.4	99.1	90.5	96.8	64.8	1.22	23.7	0.43	1.10	0.58
Bulk 4/5	26	99.5	99.4	94.0	96.1	57.4	7.8	27.4	0.46	0.96	0.49

Two POX leach tests were previously completed in the 2020 refinery program using the same POX conditions as the 2023 program. The 2020 results are presented in Table 24-5 and are consistent with the 2023 refinery program.

Table 24-5: Pressure Oxidation Leach Results Summary (2020 Program)

Test ID	Atm Residue (%feed w/w)	Leach Extraction (%)				Produced ILS (g/L)					
		Ni	Co	Cu	Mg	Ni	Fe	H_2SO_4	Cu	Co	Mg
1	N/A	99.5	99.3	92.4	89.8	69.4	2.5	21.5	1.00	1.14	0.65
2	N/A	98.8	99.4	92.9	82.8	70.1	0.77	12.1	0.97	1.08	0.62

24.2.3.6 Atmospheric Oxidation Leach

The objective of the atmospheric leach unit operation is to maximize the removal extents of sulphuric acid and iron from the ILS from the POX leach. This simultaneously achieves the following:

- purifies the leach solution to reduce the cost of downstream purification;
- effectively utilizes the contained acid in POX discharge, including acid generated through iron precipitation, to dissolve nickel;
- partially dissolves the awaruite to reduce the amount of dissolution required in the POX leach, which in turn reduces the autoclave cooling load (awaruite dissolution is exothermic); and
- increases the nickel concentration of the leach solution to reduce downstream equipment sizing (nickel sulphate solubility is greater under atmospheric leach conditions than under pressure leach conditions).

A total of fourteen batch atmospheric oxidation leach tests were completed in the 2023 refinery program. Leach tests used POX ILS and awaruite concentrate and investigated a range of reaction conditions including addition of oxidant, oxidant employed, retention time, concentrate addition rate, and ILS acidity. Optimized tests were conducted at 85°C, using oxygen, and a 0.11 concentrate to ILS ratio (w/w). Note that this concentrate to ILS ratio is approximately equivalent to a counter-current leach circuit where 100% of the fresh concentrate reports to the atmospheric leach. Results from optimized tests are presented in Table 24-6.

Table 24-6: Atmospheric Oxidation Leach Results Summary

Test ID	Res. Time (h)	Feed ILS						Produced PLS							Leach Extent	
		H_2SO_4 (g/L)	Ni (g/L)	Fe (ppm)	Cu (ppm)	Al (ppm)	Cr (ppm)	pH	ORP (mV[1])	Ni (g/L)	Fe (ppm)	Cu (ppm)	Al (ppm)	Cr (ppm)	Ni (%)	Co (%)
11	3	24	71	2,900	451	20	20	5.6	213	85	<1	<1	<3	<0.6	25	52
12	6	43	67	6,630	482	38	66	4.3	293	108	<1	151	<3	<0.6	64	89

Notes: 1. versus Ag/AgCl saturated KCl

Key conclusions from this testwork are:

- Under oxidative atmospheric leach conditions, awaruite can neutralize up to a pH greater than 5. This permits complete oxidation and precipitation of iron and complete precipitation of trace amounts of aluminum and chromium. Final concentrations of these elements in the PLS are in-line with those expected through operation of a traditional iron removal circuit, indicating that this circuit may not be required. Continuous test work is recommended to confirm the performance of the counter-current leach system.
- Precipitation of copper, aluminum, and chromium, coupled with recirculation of the atmospheric leach residue back to POX leach, creates a circulating load of these elements. Additional considerations are required to ensure these elements are removed from the counter-current leach system.
- PLS nickel concentrations greater than 100 g/L Ni are achievable.
- High dissolution extents of awaruite are achievable with the awaruite remaining sufficiently active to neutralize beyond pH 4. Note that the cobalt dissolution extent is believed to be a better indicator of awaruite dissolution than nickel since cobalt is associated with awaruite and re-precipitation of nickel as a nickel containing jarosite species was confirmed through x-ray diffraction analysis.
- The presence of excess awaruite leads to faster neutralization kinetics with complete iron removal in three hours, even under the limitations of lab-scale oxygen mass transfer rates.
- Complete copper removal is observed in Test 11 and is believed to be removed by cementation with awaruite which functions as a strong reducing agent. Complete copper removal was achieved early in Test 12, but redissolution of copper was observed at the end of the test once awaruite dissolution was near complete.

24.2.3.7 Copper Cementation

As discussed in Section 24.1.3.6, a bleed of copper from the counter-current leach system is required. Treatment of a portion of the ILS, via copper cementation with awaruite concentrate was considered and tested using ILS spiked with copper to a value representing a 10-times increase in the single pass ILS concentration. Table 24-7 summarizes the results from the cementation tests fed with ILS containing 5.1 g/L Cu, 74 g/L Ni, 3.1 g/L Fe and 22 g/L H_2SO_4 at 85°C and a 60 min residence time.

Table 24-7: Copper Cementation Results Summary

Test ID	Concentrate Addition (mol/mol[1])	Cu Removal (%)	Cu in Discharge Solution (g/L)	Copper Cement Grade (%)			
				Cu	Ni	Fe	Mg
3	0.5	43	3.06	70.5	21.1	6.3	0.1
4	1.0	84	0.83	66.0	23.9	7.3	0.1

Notes: 1. molar equivalent of Ni+Fe in awaruite / to molar equivalent of Cu and Fe^{3+} in ILS

The copper cementation tests show that high copper removal extents can be achieved. This suggests that a copper cementation circuit can be successfully implemented to remove copper from the counter-current leach. The resultant copper cement contains high copper levels with elevated nickel levels from unreacted awaruite. The nickel contained in the copper cements represents approximately 0.2% of feed to the overall circuit and further optimization testing is recommended to minimize this potential loss.

24.2.3.8 Aluminum and Chromium Removal

As discussed in Section 24.1.3.6, a bleed of aluminum and chromium from the counter-current leach system is required. A two-step Al-Cr removal process, consisting of acid dissolution of these impurities from a bleed of the atmospheric leach residue and subsequent selective Al-Cr precipitation, was considered and tested. To produce solids to model this Al-Cr removal circuit, a dedicated atmospheric leach test was conducted using POX ILS spiked to achieve Al and Cr concentrations equating to 50-times the single pass ILS concentration (1 g/L for Al and 2 g/L for Cr). The resultant solids contained 0.67% Al and 1.49% Cr and were leached with sulphuric acid at a target pH.

Initial leach tests at stoichiometric acid additions (relative to Al-Cr concentrations) yielded limited Al and Cr dissolution due to preferential dissolution of nickel. Subsequent tests, at controlled pH, were more successful in dissolving Al and Cr and are summarized in Table 24-8. High Al and Cr leach extractions were achieved creating a solids residue depleted in these elements and suitable to be recycled back to POX leach. Significant co-leaching of nickel occurred creating a solution with significantly elevated nickel concentrations.

Table 24-8: Aluminum and Chromium Removal Results Summary – Part 1 Leaching

Test ID	Target pH	Leach Extraction (%)				Residue Grade (%)				Solution Concentration (%)			
		Al	Cr	Ni	Fe	Al	Cr	Ni	Fe	Al	Cr	Ni	Fe
2	1.2	>95	85	87	92	<0.04	0.30	71.2	25.1	0.62	1.24	25.8	10.7
3	1.8	86	69	68	64	0.10	0.50	61.2	31.4	0.56	1.14	20.6	15.1

The second step of the Al and Cr removal process was selective precipitation of these elements from the first step leach solution. Precipitation extents greater than 99% for both elements were achieved when the solution pH was controlled at 4.5 with NaOH, while concurrently precipitating a moderate amount of nickel. Although this residue has a relatively high nickel grade, the low concentration of soluble Al and Cr in the awaruite concentrate results in the Al-Cr residue accounting for only 0.1% of fresh nickel feed. Additional optimization testing for both Al and Cr removal steps is recommended to reduce nickel losses and reduce the volume of the waste residue. Further, additional optimization testing is recommended to determine the optimal Al and Cr concentration in the ILS with a particular focus on the solid-liquid separation properties of the atmospheric leach residue.

Table 24-9: Aluminum and Chromium Removal Results Summary – Part 2 Precipitation

Test ID	Target pH	Precipitation Extent (%)				Residue Grade (%)				Solution Concentration (%)			
		Al	Cr	Ni	Fe	Al	Cr	Ni	Fe	Al	Cr	Ni	Fe
3	4.5	99.5	99.7	12.6	88.8	2.00	3.78	7.13	31.4	<0.003	0.003	18	1.2

24.2.3.9 Cobalt Solvent Extraction

The full composition of the PLS produced from larger-scale atmospheric leach tests and advanced to CoSX is presented in Table 24-10. The calcium, manganese, and zinc grades are low due to the low grade of these elements in the awaruite concentrate and the absence of the use of a calcium-based neutralization reagent for removal of acid and iron. This eliminates the requirement for an impurity SX step, using the extractant D2EHPA.

Table 24-10: PLS Solution Composition

Element	Al	Ca	Co	Cr	Cu	Fe	Mg	Mn	Ni	Si	Zn	pH
Grade (g/L)	<0.003	0.015	1.74	<0.0006	0.009	<0.001	0.92	0.03	83.4	0.06	0.01	5.34

A total of seven CoSX tests were completed using a phosphinic acid extractant. Testing included successive organic contacts of the PLS to produce raffinate for NiSX tests and to produce strip solution for cobalt precipitation tests. Extractions at various Aqueous:Organic (A:O) ratios were completed to generate an isotherm and engineering data.

Three successive organic contracts (10% v/v Cyanex 272, 55°C, 1:1 A:O, pH 5.0-5.3) of the PLS successfully extracted greater than 99% of the cobalt and produced a cobalt devoid raffinate with a Ni:Co ratio of greater than 1,000,000:1. These conditions resulted in co-extraction of nickel from solution (between 1 and 3%). This nickel co-extraction is expected to be significantly reduced during continuous counter-current CoSX operation including an organic scrubbing stage. Continuous CoSX testing is recommended in the next phase of study to confirm the optimal circuit configuration and conditions to reduce the nickel loss to the CoSX strip solution.

McCabe-Thiele analysis of the CoSX isotherm test data indicated that commercially, between three to four extraction stages should successfully extract greater than 99% of the cobalt and that increased A:O ratios could be employed while maintaining similarly high extraction values.

24.2.3.10 Cobalt Precipitation

Cobalt precipitation tests resulted in successful cobalt precipitation from the resultant CoSX strip solution, or synthetic equivalent. MHP tests yielded greater than 99% cobalt precipitation from solution upon pH adjustment with NaOH to above pH 8. This resulted in an MHP containing Co (38 to 40 wt%), Mg (4 wt%), Mn (0.6 wt%), Cu (0.2 wt%) and S (7 to 8 wt%).

Production of a battery-grade cobalt sulphate product was not tested, but is reasonably assumed to be technically feasible. Key considerations to achieve a high-purity cobalt sulphate product would be effective scrubbing of magnesium and nickel in CoSX and dedicated purification steps on the CoSX strip solution to remove trace levels of copper, zinc, and manganese using industrially proven ion exchange resins.

24.2.3.11 Nickel Solvent Extraction

A total of seven NiSX tests were completed using a neodecanoic acid extractant. Testing included successive organic contacts of the PLS to produce strip solution for nickel sulphate crystallization tests and to produce raffinate for nickel scavenging tests. Extractions at various A:O ratios were completed to generate an isotherm and engineering data.

NiSX single stage extraction (20%v/v Versatic 10, A:O 0.2, 55°C, pH 6.3-6.5) achieved approximately 80% nickel extraction. McCabe-Thiele analysis of the isotherm test data indicated that four extraction stages would be required to yield nickel extraction greater than 98%. Loaded organic was washed with water to remove entrained soluble impurities. Single-stage acid stripping resulted in partial nickel removal to produce a concentrated nickel solution with low free acidity for nickel sulphate crystallization testing. Multiple acid strip stages are required commercially to achieve complete nickel removal.

Magnesium concentrations in the NiSX feed solution were below those expected during industrial operation due to the batch manner of CoSX testing (which increases magnesium extraction) and the unrepresentatively low magnesium grade in the tested awaruite concentrate. An additional isotherm test series was conducted to evaluate the impact of higher magnesium concentrations using NiSX feed spiked with magnesium sulphate. Nickel extractions were not materially impacted by the increased magnesium concentration, but increased magnesium extraction rates were observed as expected. At the highest A:O ratio, 0.25, the ratio of loaded Ni to Mg was approximately 2,300 (at pH 6.7). This is approximately 20 times lower than the ratio required in the NiSX strip solution to meet a 5 ppm Mg specification in the nickel sulphate crystals. This highlights the need for careful selection of continuous extraction and scrub conditions to ensure acceptable magnesium rejection. Continuous NiSX testing is recommended in the next phase of study to confirm the circuit configuration and conditions required for acceptable magnesium rejection. This could be extended to partial magnesium extraction in CoSX to limit the magnesium concentration feeding NiSX.

24.2.3.12 Nickel Sulphate Crystallization

Strip solution from NiSX was used in batch crystallization tests. A crystallization temperature of 50°C was used to ensure formation of alpha nickel sulphate hexahydrate, the preferred form of nickel sulphate for battery applications. Four successive crops of nickel sulphate crystals were obtained in a semi-continuous manner and the assays for the washed crystals are presented in Table 24-11, along with the target specification.

The first crop meets all specifications. The subsequent three crops contain elevated impurities of either silicon or iron. External contamination of these crystal samples is strongly suspected due to the absence of these elements from the feed solution and crystallization of mother liquor. Potassium levels are all below the target specification but are suspected to be anomalously high for the same reasons. Zinc concentrations are below the target specification but are higher than expected as near complete zinc removal was observed in CoSX. It is not evident if the zinc in the nickel sulphate crystals was due to contamination of the CoSX raffinate or some other transfer mechanism. Further investigations into this are recommended in the next phase of study. If required, removal of zinc from the NiSX strip solution is expected to be technically feasible using a D2EPHA impregnated ion exchange resin.

Table 24-11: Nickel Sulphate Crystallization Results Summary

Element	Unit	Crop 1	Crop 2	Crop 3	Crop 4	Target Specification
Nickel (Ni)	Percentage	22.0	22.9	22.5	22.6	>22
Sulphur (S)	Percentage	12.0	12.1	11.9	12.2	N/A
Aluminum (Al)	ppm	<0.05	<0.05	<0.05	0.2	<5
Arsenic (As)	ppm	0.5[1]	0.6[1]	0.6[1]	0.4[1]	<2
Calcium (Ca)	ppm	<0.5	<0.5	<0.5	3.0	<5
Cadmium (Cd)	ppm	<0.05	<0.05	<0.05	<0.05	<1
Cobalt (Co)	ppm	<0.05	0.96	0.96	1.90	<50
Chromium (Cr)	ppm	<0.05	<0.05	<0.05	0.08	<3
Copeer (Cu)	ppm	0.08	0.10	0.1	0.3	<3
Iron (Fe)	ppm	0.5	1.0	0.9	57.9[1]	<3
Potassium (K)	ppm	3.0	7.8	3.0	<0.5	<10
Magnesium (Mg)	ppm	<0.5	<0.5	<0.5	2.0	<5
Manganese (Mn)	ppm	<0.05	0.1	0.4	0.5	<5
Sodium (Na)	ppm	2	5.5	8.8	14.0	<20
Lead (Pb)	ppm	<0.05	<0.05	<0.05	<0.05	<2
Silicon (Si)	ppm	<2.0	488[1]	35[1]	46.8[1]	<10
Zinc (Zn)	ppm	2	2.4	3.9	3.9	<5

Notes: 1. External sample contamination strongly suspected due to absence of these elements from the feed solution and mother liquor

24.2.3.13 Nickel Scavenging Precipitation

Five nickel scavenging tests were completed to recover residual nickel from the NiSX raffinate. Key results from precipitation tests conducted at 60°C on NiSX raffinate grading 0.30 g/L Ni and 0.94 g/L Mg are summarized in Table 24-12. The results demonstrate good nickel precipitation selectivity over magnesium, indicating complete nickel recovery with a manageable co-precipitation of magnesium. Re-leach of the precipitate, to recover nickel values, was not tested but can be reasonably expected to be near complete.

Table 24-12: Nickel Scavenging Results Summary

Test ID	NaOH (mol/mol[1])	Precipitation Extent (%)		Discharge Solution (g/L)	
		Ni	Mg	Ni	Mg
3	3	64.8	2.6	0.11	0.92
4	4	>99.0	5.9	<0.003	0.89

Notes: 1. molar equivalent of NaOH / to molar equivalent of Ni in raffinate

24.2.3.14 Magnesium Removal

No magnesium removal testing on the nickel scavenging discharge solution was completed. Removal of magnesium down to <0.05 g/L through hydroxide precipitation is well established industrially and the exact performance of this unit operation has limited impacts on the overall circuit.

24.2.3.15 Sodium Sulphate Crystallization

No sodium sulphate crystallization removal testing was completed. Crystallization of anhydrous sodium sulphate is well established industrially. The purity of the sodium sulphate produced was not determined but is expected to be high due to the low concentrations of any other ions in solution.

24.2.3.16 Product Quality

24.2.3.16.1 Nickel Sulphate Crystals

Nickel sulphate crystal assays produced from testwork were previously presented in Table 24-11. As there is no agreed commercial specification of nickel sulphate for battery applications and different end-users have different and evolving requirements, the target specification for “battery-grade” is based on a survey of available high purity nickel sulphate specifications. Consistency of product quality is a major requirement for battery-grade products though this is related to industrial design of the nickel sulphate refinery and cannot be properly assessed from the bench-scale test program.

The crystals produced from awaruite concentrate meet or exceed the quality requirements for all elements in the target nickel sulphate specification. This indicates the suitability of the awaruite concentrate from Baptiste as feedstock to produce nickel sulphate specifications for EV battery applications.

24.2.3.16.2 Cobalt Hydroxide

Cobalt hydroxide produced from testwork ranges from 38 to 40% Co, 4% Mg, 0.6% Mn, Cu 0.2 % and 7 to 8% S. This is a higher quality cobalt hydroxide product (most notably lower in Mn) than those typically produced from mining operations in the Central African Copperbelt. The Baptiste cobalt hydroxide is thus expected to be readily salable into refineries that treat these or similar feedstocks at comparable or improved payability.

24.2.3.16.3 Copper Cement

Copper cement produced from testwork ranges from 66 to 70% Cu, 21 to 24% Ni, 6 to 7% Fe, and 0.1% Mg. The elevated nickel concentrations in the copper cement indicate this material would most suitably be sold to a refinery designed to treat copper-rich nickel matte including base metal refineries associated with PGM operations. Copper and nickel values can be reasonably expected to be payable while the small amount of magnesium is expected to be a penalty element.

24.2.3.17 Recovery Model

Estimates of the overall circuit recovery have been made based on the presented testwork, industrial benchmarking and circuit heat and mass balance modelling. The recoveries assumed in the economic analysis based for each element are:

- nickel recovery of 99.2%;
- cobalt recovery 96.3%; and
- copper recovery of 94.9%.

24.2.4 Mineral Resource Estimates

Refer to Section 14 as the Refinery Option has no effect on resource estimate.

24.2.5 Mineral Reserve Estimates

Refer to Section 15 as the Refinery Option has no effect on reserve estimate.

24.2.6 Mining Methods

Refer to Section 16 as the Refinery Option has no effect on mining. The Refinery Option is designed to process awaruite and MHP concentrate produced by the Baptiste concentrator and will not influence mining.

24.2.7 Recovery Methods

Refer to Section 17 for the Baptiste concentrator, as the Refinery Option has no effect on the concentrator.

The proposed refinery will process awaruite concentrate and MHP from the Baptiste concentrator and produce primarily nickel sulphate as well as cobalt hydroxide, copper cement, and sodium sulphate. The refinery is designed for an output of 40 kt/a of nickel in nickel sulphate. The refinery will be fed predominantly with the awaruite concentrate from the Baptiste concentrator. The MHP produced from the Baptiste concentrator is also fed to the refinery and represents 7.5% of the feed nickel units.

An overarching criterion for the design of the refinery was to be a calcium 'free' system, which simplifies the impurity removal and reduces complexity around managing calcium saturated solution in solvent extraction. Therefore, a design consideration for feeding MHP to the refinery is the presence of calcium. Calcium is not present in material quantities in the awaruite concentrate so there was no need to have a calcium removal stage. Considering the low calcium grade in the MHP (0.75%) and the small percentage of the total refinery feed it would make, resulting calcium concentration in the PLS do not warrant changing the PLS purification scheme which was designed for only refining awaruite concentrate.

The following major process steps included in the proposed refinery flowsheet are:

- awaruite concentrate and MHP solids repulp;
- atmospheric leach;
- pressure leach;
- aluminum and chromium removal;
- copper cementation;
- cobalt solvent extraction and cobalt hydroxide precipitation;
- nickel solvent extraction and nickel sulphate crystallization;
- magnesium precipitation; and
- sodium sulphate crystallization.

24.2.7.1 Plant Design

Key process design criteria for the refinery are listed in Table 24-13.

Table 24-13: Summary of Major Processing Design Criteria – Refinery Option

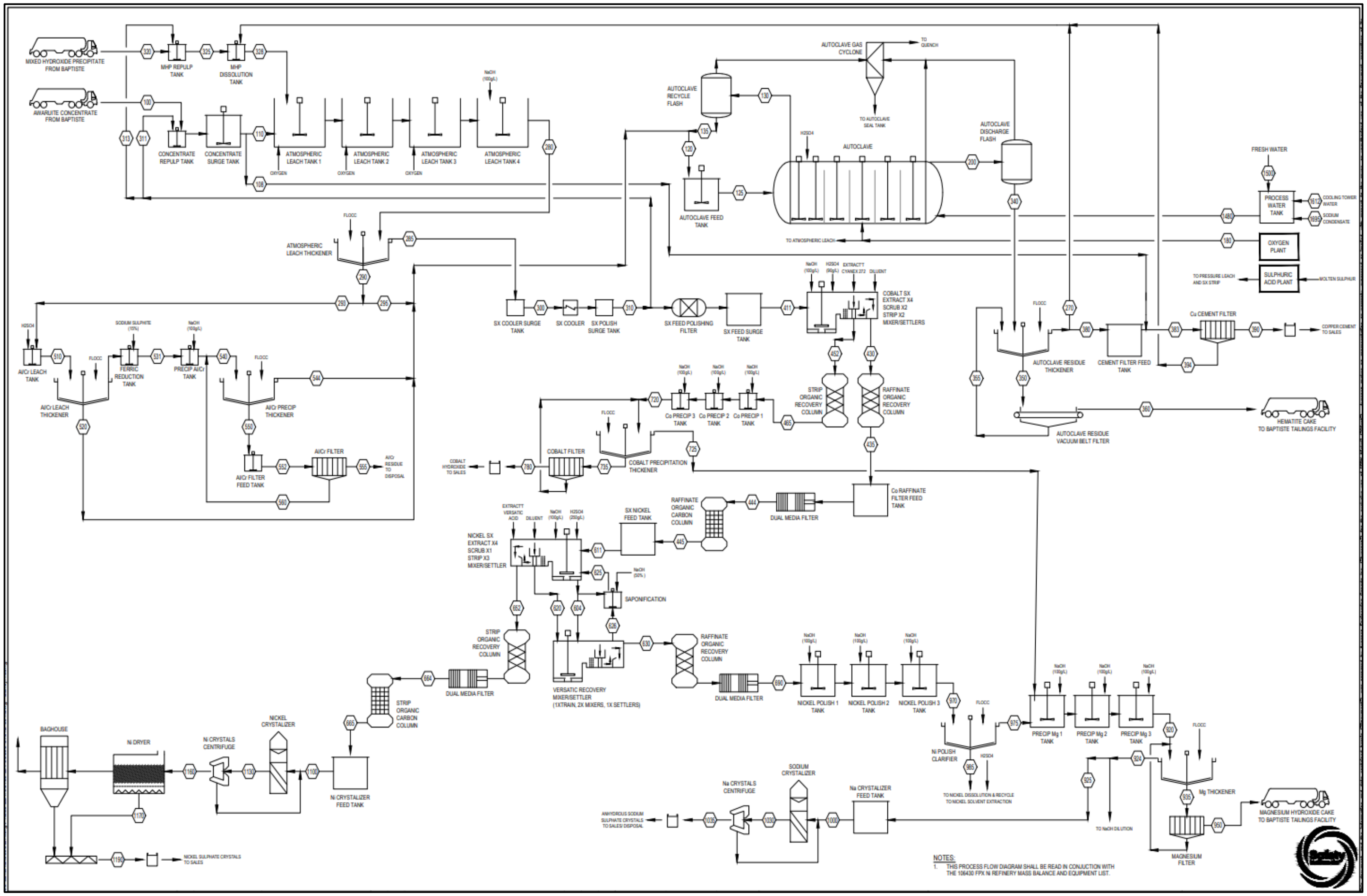
Criteria	Unit	Value
Design Capacity	t/a Nickel in Product	40,000
Awaruite Concentrate Feed	t/h	7.15
MHP Concentrate Feed	t/h	0.75
Refinery Availability	%	92
Filtration Availability	%	85
Awaruite Concentrate Feed (Design)		
Ni	%	65.0
Fe	%	24.0
Co	%	1.1
Mg	%	1.0
Cu	%	0.5
MHP Concentrate Feed (Design)		
Ni	%	50.0
Mg	%	1.7
Ca	%	0.8

Criteria	Unit	Value
Nickel Recovery in Nickel Sulphate Crystal (as $NiSO_4.6H_2O$)	%	99.2
Co Recovery in Cobalt Hydroxide	%	96.3
Cu Recovery in Copper Cement	%	94.9
Autoclave Retention	min	60
Autoclave Discharge Acid Concentration	g/L H_2SO_4	25
Atmospheric Leach Retention	min	240
Atmospheric Leach Discharge pH	Unit	5.0-5.2
Solvent Extraction Nickel Concentration	g/L	100-105

24.2.7.2 Process Flow Sheet

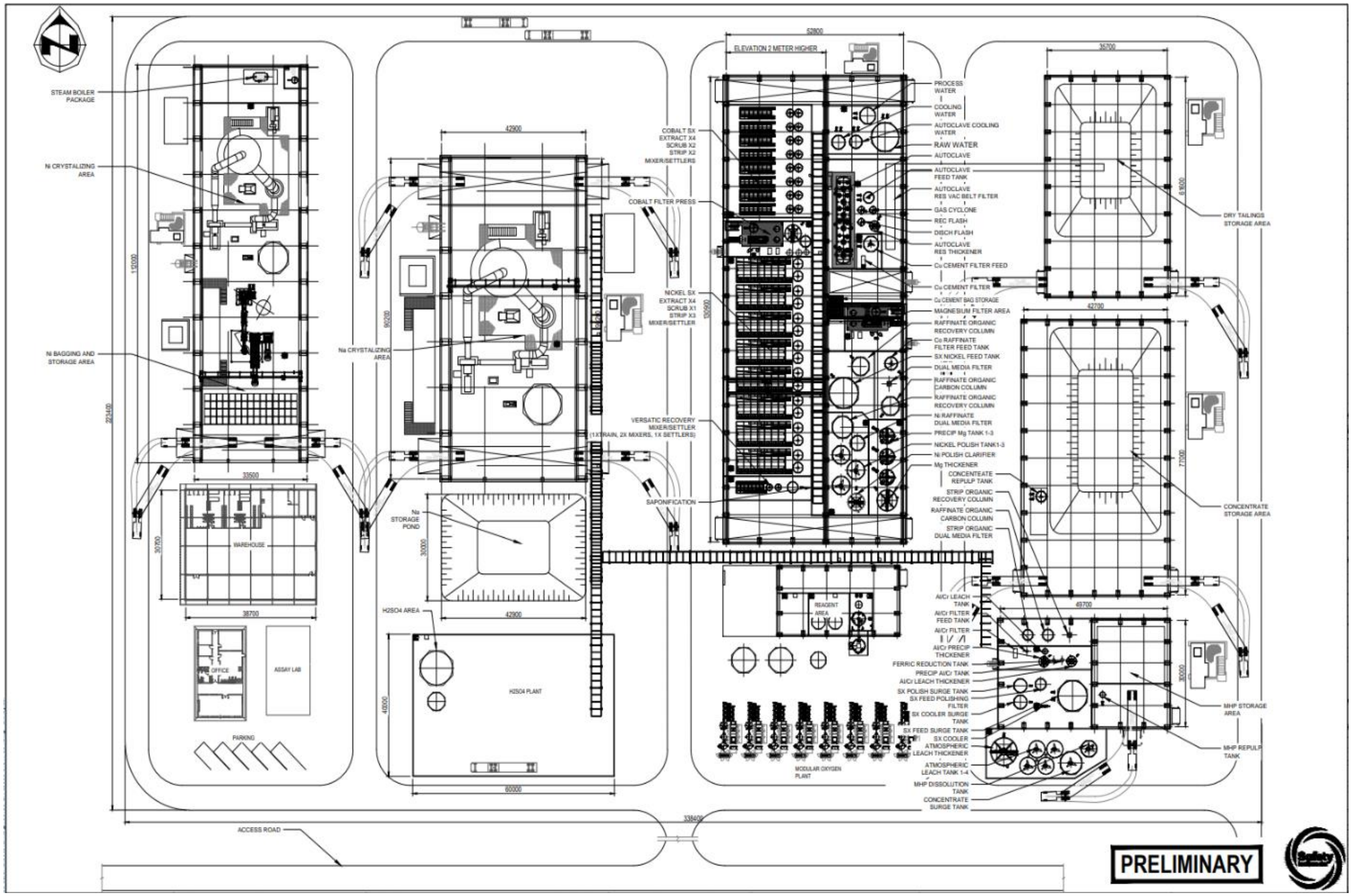
The refinery will have an availability of 92%. Details of the refinery flow sheet are described in the following sections. The refinery overall process flow diagram is provided in Figure 24-4. The overall general arrangement of the refinery can be found in Figure 24-5.

Figure 24-4: Overall Process Flow Diagram – Refinery Option



Source: Ausenco, 2023.

Figure 24-5: Overall Site Layout – Refinery Option



Source: Ausenco, 2023.

24.2.7.2.1 Feed Preparation and Atmospheric Leaching

MHP and awaruite concentrate from the Baptiste concentrator will be delivered to site and fed to the refinery at a rate of 0.75 t/h and 7.15 t/h, respectively. The two feeds will be repulped in separate repulping tanks prior to being fed to atmospheric leaching.

The MHP will require an additional dissolution tank prior to leaching. The dissolved MHP and awaruite pulp will be combined and leached in a series of four atmospheric leach tanks with acid recycled from the POX leach and oxygen sparged into tanks one through three. In the atmospheric leach the acid from the pressure oxidation is consumed leaving a solution containing 90-100 g/L nickel at a pH of 5. The intent is not to leach all the nickel in this step, but to consume acid and precipitate iron with awaruite concentrate.

The discharge from the leach reactors will be sent to the atmospheric leach thickener. The thickener will overflow to the cobalt solvent extraction feed preparation circuit and the underflow, thickened to 40% pulp density by weight, will be sent to the autoclave to leach the remaining nickel. A small fraction of the atmospheric leach thickener underflow (2 - 3%) will proceed to the aluminum-chromium removal circuit.

In the atmospheric leach, awaruite is leached with acid and oxygen, while copper, aluminum and chromium leached during the pressure oxidation step are precipitated from solution, necessitating the need for these elements to be recovered in small secondary processes. The main reactions are:

- $4H_2SO_4(a) + Ni_3Fe(s) + 2O_2(g) \rightarrow 3NiSO_4(a) + FeSO_4(a) + 4H_2O(a)$
- $4CuSO_4(a) + Ni_3Fe(s) \rightarrow 4Cu(s) + 3NiSO_4(a) + FeSO_4(a)$
- $2FeSO_4(a) + O_2(g) + 2H_2O(a) \rightarrow 2FeOOH(s) + 2H_2SO_4(a)$
- $Al_2(SO_4)_3(a) + 6H_2O(a) \rightarrow 2Al(OH)_3(a) + 3H_2SO_4(a)$
- $Cr_2(SO_4)_3(a) + 6H_2O(a) \rightarrow 2Cr(OH)_3(a) + 3H_2SO_4(a)$

The area will include the following major equipment:

- Four atmospheric leach tanks, 85 m^3 each.
- Atmospheric leach thickener, 8 m diameter.
- Associated pumps.

24.2.7.2.2 Aluminum/Chromium Bleed Treatment

To prevent build up in the leaching circuit, the aluminum and chromium precipitated during the atmospheric leach must be removed via a bleed. To achieve this, 2-3% of the atmospheric leach thickener underflow will be leached with sulphuric acid to dissolve primarily aluminum and chromium, but also iron and other metals that precipitated in the atmospheric leach. The leach slurry is thickened, with the nickel rich thickener underflow sent to the pressure leach circuit while the overflow will be treated with sodium hydroxide to precipitate the leached aluminum and chromium.

The precipitated solids will be thickened in the Al/Cr precipitation thickener and the underflow filtered in the Al/Cr filter press. The filter press filtrate will be recycled into the Al/Cr precipitation thickener and the filter cake sent off-site for appropriate disposal. Overflow from the Al/Cr precipitation thickener will be sent to the pressure leach circuit. The area will include the following equipment:

- Al/Cr leach thickener, 3 m diameter.
- Al/Cr precipitation thickener, 3 m diameter.
- Al/Cr filter, 8.6 m^2 total filtration area.
- Al/Cr leach tank, 3 m^3.
- Associated pumps.

24.2.7.2.3 Pressure Leach

The partially leached awaruite concentrate from the atmospheric leach thickener underflow, Al/Cr leach thickener underflow and Al/Cr precipitation thickener overflow will be combined and fed into the autoclave with sulphuric acid (to a target of 25 g/L in the discharge) and oxygen sparged to each compartment. The nickel in the awaruite is leached into solution while the iron is oxidized and precipitated as hematite. The main reactions are:

- $Ni_3Fe(s) + 4H_2SO_4(a) + 2O_2(g) \rightarrow 3NiSO_4(a) + Fe2SO_4(a) + 4H_2O(a)$
- $4FeSO_4(a) + O_2(g) + 4H_2O(a) \rightarrow 2Fe_2O_3(s) + 4H_2SO_4(a)$

Along with nickel, cobalt, copper, chromium, aluminum and magnesium are also leached into solution.

The pressure leach will occur in a five-compartment autoclave, operated at a temperature of 150°C and a pressure of 930 kPa, with a total retention time of 1 hour. The circuit will be equipped with a flash recycle system, discharge flash, and gas cyclone.

The autoclave discharge will enter the flash vessel where the pressure-drop flashes water to steam. The vapor will go to the autoclave gas cyclone and the slurry will be pumped to the autoclave residue thickener.

The underflow from the autoclave residue thickener will be further dewatered and washed on the autoclave residue vacuum belt filter with the leach residue stored for eventual backhaul to the tailings facility at the Baptiste site. The filter cake will be washed on the belt filter with process water to recover soluble nickel values. Filtrate and wash from the belt filter are recycled into the autoclave residue thickener.

The overflow from the autoclave residue thickener will be sent to the MHP dissolution tank with a bleed stream being sent to copper cementation.

Oxygen will be supplied at 1,000 kPa from the on-site oxygen plant. Cooling water for the autoclave will be drawn from the process water tank.

The area will include the following major equipment:

- Five compartment autoclave, 77.5 m^3 total compartment volume for 1 hour retention, including flash recycle, discharge flash, and gas cyclone.
- Autoclave residue thickener, 5 m diameter.

- Autoclave residue vacuum belt filter, 30 m^2 belt area.
- Associated pumps.

24.2.7.2.4 Copper Cementation

Twenty percent of the overflow from the autoclave residue thickener is treated with awaruite concentrate to remove copper from the leach circuit. Fifty percent of the soluble copper is cemented with awaruite concentrate sourced from the concentrate surge tank. Maintaining a reasonably high residual copper tenor in the solution allows the copper grade in the cement product to be maximized, and will minimize the associated nickel losses in the solid product. The bleed solution and the concentrate will be mixed and then recovered on a pressure filter in a process similar to Merrill-Crowe.

$Ni_3Fe(s) + 4CuSO_4(a) \rightarrow 4Cu(s) + 3NiSO_4(a) + FeSO_4(a)$

The filter cake will be the final copper cement product and will be stored and shipped off site for sale. The filter filtrate and wash is forwarded to the atmospheric leach via the MHP dissolution tank.

The area will include the following equipment:

- Copper cement filter, 14.4 m^2 total filtration area.
- Associated pumps.

24.2.7.2.5 Cobalt Solvent Extraction and Precipitation

Overflow from the atmospheric leach thickener will be cooled in the solvent extraction cooler before entering the solvent extraction polishing filter to remove any remaining suspended solids. A portion of the treated solution is recycled to the awaruite concentrate and MHP repulp tanks in the leach circuit to build the nickel concentration in the solvent extraction feed solution to 100 g/L. Cobalt in the solution builds to around 1.4 g/L.

The cobalt solvent extraction circuit consists of a series of eight total mixer settlers, with four for extract, two for scrub, and two for strip. The extract stage contacts the leach solution and the Cyanex 272 (or equivalent) organic phase in counter current flow to transfer the cobalt from the aqueous onto the organic phase. To ensure high cobalt recovery, the pH in each extraction step is controlled at pH 5 using dilute sodium hydroxide. Overall, the amount of caustic added is determined by the amount of cobalt being extracted per the following reaction.

$CoSO_4(a) + 2LIX\text{-}H(o) + 2NaOH(a) \rightarrow (LIX)_2\text{-}Co(o) + Na_2SO_4(a) + 2H_2O(a)$

The cobalt loaded organic from the extract stages is contacted with a cobalt solution (from strip) in the two-stage scrub circuit to remove co-extracted nickel, magnesium and calcium from the organic. The aqueous scrub solution, containing the removed impurities, is returned to extraction to recover the cobalt, while the purified organic phase is contacted with a high acidic strip solution to remove the cobalt from the organic and into the strip solution. A 90 g/L acid solution in scrub will produce a strip solution with around 50 g/L cobalt and 5 g/L free acid; a portion of this solution is used for scrubbing.

The strip solution goes through a multi-stage organic recovery system to minimize organic losses and prevent contamination of the cobalt product. The clean solution is treated with sodium hydroxide in a three-tank plant to precipitate cobalt hydroxide.

$CoSO_4(a) + 2NaOH(a) \rightarrow Co(OH)_2(a) + Na_2SO_4(a)$

The cobalt precipitate is thickened to 32% by weight in the underflow. A portion is recycled to provide seed for precipitation, with the remainder further dewatered to 60% solids by weight in the cobalt filter prior to packaging and shipping off site for sale. Filtrate and wash solution are returned to the cobalt thickener and form part of the overflow which is sent to the magnesium precipitation tanks.

The nickel bearing raffinate from the cobalt solvent extraction process will be pumped through an organic recovery column and a dual media filter to recover entrained organic prior to entering nickel solvent extraction. This step is critical to avoid cross contamination of organic extractants and maintain process selectivity.

The area will include the following major equipment:

- Solvent extraction polishing filter, 57 m^3/h throughput.
- One 450 m^3 solvent extraction surge tank.
- Cobalt solvent extraction, eight mixer settlers, 4-2-2 (extract, scrub, and strip).
- Various solvent extraction tanks for loaded organic, raffinate, strip and scrub solution.
- Organic recovery columns and media filters.
- Three 4 m^3 cobalt precipitation tanks, thickener, and filter.
- Associated pumps.

24.2.7.2.6 Nickel Solvent Extraction and Crystallization

The nickel solvent extraction circuit will consist of a series of eight total mixer settlers, with four for extract, one for scrub, and three for strip, and will operate in a similar manner as the cobalt SX but uses versatic acid as the organic extractant.

Due to the high concentration of nickel in solution and therefore the high demand for sodium hydroxide for pH control in extraction, saponification is used to pretreat the stripped organic before extraction. In saponification, the organic is contacted with concentrated sodium hydroxide solution which reduced the final solution volume that needs to be treated through the sodium sulphate crystallizer. In the refinery design, 70% of the nickel equivalent neutralization is achieved using a 50% caustic solution (in saponification) with the remaining 30% nickel equivalent neutralization using the dilute 100 g/L (8.3% by weight) solution in the extraction mixers. In saponification, the sodium loads onto the reagent and then exchanges for nickel in extraction.

$Ver\text{-}H(o) + NaOH(a) \rightarrow Ver\text{-}Na(o) + H_2O(a)$ saponification

$NiSO_4(a) + 2Ver\text{-}Na(o) \rightarrow (Ver)_2\text{-}Ni(o) + Na_2SO_4(a)$ extraction

$NiSO_4(a) + 2Ver\text{-}H(o) + 2NaOH(a) \rightarrow (Ver)_2\text{-}Ni(o) + Na_2SO_4(a) + 2H_2O(a)$ extraction

The final pH for nickel extraction with versatic acid will be around 6.5. At this pH, versatic is slightly soluble in aqueous solutions, so raffinate from the extraction stage is mixed with organic for the strip stage and sulphuric acid to recover versatic acid by reducing its solubility in the aqueous phase. This is completed in a single mixer-settler stage.

After versatic recovery, the NiSX raffinate will pass through a multi-stage organic recovery process prior to feeding the nickel polishing circuit. Any residual nickel in the raffinate is precipitated in three tanks, using sodium hydroxide. The solution will be clarified in the nickel polish clarifier, with the underflow reporting back to the nickel solvent extraction circuit via a nickel dissolution tank. The overflow will be sent for magnesium and sodium removal. Further testing may show the nickel concentration in the NiSX raffinate can be maintained at a low enough concentration that this step is not required.

The nickel loaded organic phase is scrubbed to remove co-extracted cobalt, magnesium and calcium using a scrub solution prepared by diluting strip solution. The clean organic will be stripped using 250 g/L sulphuric acid to generate a near saturated nickel sulphate solution with 160 g/L nickel and 5 g/L free acid. The nickel strip solution will pass through the strip organic recovery column followed by a dual media filter and a carbon column to remove entrained organic.

Once clean of organics, the strip solution will be sent to nickel crystallization. The solution will be preheated before entering the crystallizer. The crystallizer will operate under vacuum to maintain a temperature of around 50°C to produce the preferred α-$NiSO_4.6H_2O$ crystal product. A high circulating rate will maintain around 15-20 % solids in the system and provide surface area for crystal growth while preventing the excessive temperature rise across the heat exchanger. Power for the system will be added via the steam compressor which uses mechanical vapour recompression (MVR) to reduce energy demand.

The crystals will be pumped out of the crystallizer and then dewatered in a centrifuge, with the centrate recycling into the centrate tank for reuse in the crystallizer. Crystals will be washed on the centrifuge before being dried and bagged for storage and shipment off site for sale.

The area will include the following major equipment:

- Nickel solvent extraction, 4-1-3 (extract, scrub, and strip).
- Versatic recovery mixer-settler.
- Various solvent extraction tanks for loaded organic, raffinate, strip and scrub solution.
- Organic recovery columns and media filters.
- Three nickel polish tanks, 100 m^3 each.
- Nickel polish clarifier, 8 m^2 diameter.
- Nickel crystallization package (pre-heater, crystallizer, centrifuge, and MVR steam compressor).
- Nickel crystals dryer and baghouse.
- Nickel crystal bagging plant.
- Associated pumps, dual media filters, recovery columns.

24.2.7.2.7 Magnesium Removal

Overflow from the nickel polish clarifier and cobalt precipitation thickener will be mixed with sodium hydroxide in a series of three tanks to precipitate magnesium hydroxide. The magnesium thickener will thicken the precipitate which will be further dewatered in the magnesium filter press. The magnesium hydroxide filter cake is stored before back hauling for disposal in the tailings facility. The magnesium thickener overflow will be used to prepare the 100 g/L dilute caustic solution, with the excess feeding of the sodium crystallizer.

The area will include the following major equipment:

- Three magnesium precipitation tanks, 61 m^3 each.
- Magnesium thickener, 9 m diameter.
- Magnesium precipitate filter press, 18 m^2.
- Associated pumps.

24.2.7.2.8 Sodium Crystallization

One of the main considerations in the design of the refinery was to minimize the evaporation load in the sodium crystallizer by recycling solutions and minimizing water addition as much as possible. Resultantly, the solution feeding sodium crystallization is at 60 g/L.

The sodium crystallizer operates in the same way as the nickel sulphate crystallizer, except the temperature is higher to generate an anhydrous Na_2SO_4 crystal product. Sodium sulphate crystals formed in the crystallizer are dewatered in the centrifuge. The sodium sulphate crystals will be dried, bagged, and free-issued to regional kraft pulp mills.

The area will include the following major equipment:

- Sodium crystallization package (pre-heater, crystallizer, centrifuge, and MVR steam compressor).
- Sodium sulphate crystals dryer and baghouse.
- Associated pumps.

24.2.7.2.9 Oxygen Plant

An oxygen plant will be constructed and operated with the refinery to supply gaseous oxygen to the POX, atmospheric leaching circuit, and sulphuric acid plant. The oxygen plant will utilize Vacuum Swing Adsorption (VSA) technology to produce low pressure oxygen at a purity of 96%. A combined 384 t/d of oxygen will be generated by the oxygen plant. Supply to the autoclave will require an additional booster system to reach the supply pressure target of 1,000 kPa(a).

24.2.7.2.10 Sulphuric Acid Plant

The sulphuric acid plant will be designed to produce 625 t/d, enough sulphuric acid for both the operations at the refinery and concentrator. Molten sulphur will be fed to the sulphuric acid plant to produce a 98% purity sulphuric acid.

24.2.7.3 Reagents

The following list details the reagents used in the refining process:

- Oxygen

Gaseous oxygen will be produced on site and used in the production of sulphuric acid and in the two leaching circuits. 384 t/d of 96% purity oxygen will be produced.

- Molten Sulphur

  Molten sulphur will be used to produce sulphuric acid. 200 t/d of molten sulphur will be supplied to the refinery to produce the 625 t/d sulphuric acid required for both the refinery and concentrator.

- Sulphuric Acid

  Sulphuric acid is generated on-site using a molten sulphur feed to the sulphuric acid plant. 625 t/d of 98% purity sulphuric acid is produced and consumed each day between the refinery (450 t/d) and the Baptiste concentrator (175 t/d). Sulphuric acid is used at three different concentrations, 98%, 250 g/L and 90 g/L depending on the use within the refinery.

- Sodium Hydroxide (NaOH)

  400 t/d of sodium hydroxide sodium (50%w/w) will be consumed for the solvent extraction processes, cobalt precipitation, nickel polish, and magnesium precipitation. Eighty percent of the caustic is used undiluted in the nickel solvent extraction, with the remainder diluted to 100 g/L for use across the refinery.

- Shellsol 2046 (Solvent Extraction Diluent)

  Shellsol 2046 is the diluent used in the solvent extraction for both nickel and cobalt.

- Cyanex 272 (Cobalt Extractant)

  Cyanex 272 is the extractant used in the recovery of cobalt in the cobalt solvent extraction circuit.

- Versatic 10 (Nickel Extractant)

  Versatic 10 is the extractant used in the recovery of nickel in the nickel solvent extraction circuit.

- Flocculant

  Flocculant will be added to each of the thickeners used in the refinery.

24.2.8 Project Infrastructure

At the current level of study, the exact location for the refinery and relevant facilities have not yet been confirmed. It is reasonably assumed that a city like Prince George will have the infrastructure and utilities to sustain a refinery of this throughput. By locating the refinery in an established city, it is expected that labour, road infrastructure, communications, water, power, operating consumables, and medical care are readily available.

24.2.8.1 Buildings

The refinery and support infrastructure are detailed in the following subsections. Table 24-13 breaks down the footprint of the refinery buildings and other support infrastructure. The total footprint of refinery buildings will be approximately 75,300 m^2.

Table 24-14: Building Footprint by Area – Refinery Option

Footprint	Width (m)	Length (m)
Main Refinery	53	131
Feed Preparation	30	50
Nickel Crystallization	34	112
Sodium Crystallization	43	91
Sodium Storage Pond	30	46
Warehouse	31	39
Office	13	25
Assay Lab	12	24
Sulphuric Acid Plant	40	60
Oxygen Plant	40	60
Reagent Storage	20	34
Dry Tailings Storage	36	62
Concentrate Storage	43	77

Source: Ausenco, 2023.

24.2.8.2 Roads and Logistics

The refinery will use public roads which are part of the chosen municipality to transport materials and people to and from site, as well as for product transportation to the rail depot. Acid produced in the refinery acid plant will be transported by public roads to Baptiste for use in the concentrator.

24.2.8.3 Power Supply

Prince George has an established power grid serviced by the provincial electrical power utilities company, BC Hydro. BC Hydro has an extensive power grid set up across municipalities in BC with a sufficient supply of stable power to meet the Refinery Option’s relatively modest power demands.

A 20 MVA transformer will be used to step down the supply voltage from 69 kV (assumed) to 4.16 kV to power the oxygen plant, sulphuric acid plant, and compressors. The voltage will be stepped down once more from this 4.16 kV bus to 600 V via two 2.5 MVA transformers to supply low voltage power to the refinery. The refinery will have a total installed power of 18 MW with an annual electrical power consumption of 111.8 GWh/a under typical operating parameters.

There will also be a single 500 kW standby diesel generator to provide emergency backup power. Upon a power outage, the supply to the essential loads will be transferred by an automatic transfer switch (ATS) to this generator.

24.2.8.4 Water Supply

It is assumed that fresh water will be supplied via the chosen area’s existing water infrastructure. The refinery will also utilize the public water and sewer systems to remove excess water and sewage. Fresh water consumption of the refinery will be approximately 16 m^3/h.

24.2.8.5 Logistics requirements

Leach residue and magnesium tailings will be temporarily stored in a dry tailings storage area before being back-hauled to the Baptiste tailings facility for disposal. The combined production rate of dry tailings is 4 t/h.

The aluminum and chromium residue filter cake will require an appropriate waste disposal facility and will be unfit for disposal in the Baptiste tailings facility. Special care will be taken to store, transport, and dispose of the aluminum and chromium residue at a specialized hazardous waste disposal facility. There will be 130 t of this waste required to be managed and disposed of each year.

24.2.9 Market Studies and Contracts

The chemistry of the refinery nickel sulphate crystal, as achieved during PFS metallurgical testwork, is presented in Table 24-15, along with an amalgamation of typical target specifications as published by major battery manufacturers. As clearly shown, the Baptiste product meets or exceeds typical target specifications for all elements of interest.

Table 24-15: Baptiste Nickel Sulphate Crystal Quality vs. Typical Target Specification

Element	Units	Nickel Sulphate Crystals	
		Baptiste Testwork	Target Specification[1]
Nickel (Ni)	wt%	>22	>22
Aluminum (Al)	ppm	<1	<5
Arsenic (As)	ppm	<1	<2
Calcium (Ca)	ppm	<1	<5
Cadmium (Cd)	ppm	<1	<1
Cobalt (Co)	ppm	1	<50
Chromium (Cr)	ppm	<1	<3
Copper (Cu)	ppm	<1	<3
Iron (Fe)	ppm	1	<3
Potassium (K)	ppm	3	<10
Magnesium (Mg)	ppm	<1	<5
Manganese (Mn)	ppm	<1	<5
Sodium (Na)	ppm	2	<20
Lead (Pb)	ppm	<1	<2
Silicon (Si)	ppm	<2	<10
Zinc (Zn)	ppm	2	<5

Notes: 1. Target specification set by FPX based on benchmarking of publicly available nickel sulphate specifications.

Based on 2018 to 2022 market pricing data published by Morgan Stanley, the realized nickel sulphate premium and discount has been unstable, ranging from a significant discount (-40%) to a significant premium (+40%) to the LME nickel price. This is demonstrative of the currently small size and inherent volatility of the nickel sulphate market, which is further amplified by volatility of the overall nickel market during the same period, as demonstrated during the extreme spike in the LME price in early 2022. Based on market data published by Asian Metal, the 2022 nickel sulphate price in China ranged from a low of $23,677 to a high of $33,036/t of contained nickel.

As the nickel sulphate market grows in the coming years and preferred feedstocks are established, it is expected that a more consistent premium basis will be established based on typical upgrading costs. For the Baptiste nickel sulphate product, a premium of $1.00/lb ($2,205/t) to the assumed base LME nickel price of $8.75/lb ($19,290/t) has been applied in the Refinery Option economic analysis. This represents a realized price of $21,495/t of contained nickel, or an approximate 11% premium over the assumed base LME nickel price, or 17% more than the FeNi product in the PFS Base Case. The QP for this section has reviewed the studies and analyses and confirms that the results support the assumptions in Section 24.2.12.

For discussion on the pricing basis for the balance of Baptiste awaruite concentrate production which will continue to be directed to the stainless-steel market as per the PFS Base Case, see Section 17.

24.2.10 Environmental Studies, Permitting, and Social and Community Impact

24.2.10.1 Environmental Studies

The Refinery Option site selection alternatives study and the selected location will be subject to environmental characterization studies. The environmental studies will be defined in the Project EA AIR document. Siting of the refinery within or adjacent to municipal boundaries is likely to require additional emphasis on certain studies such as evaluation of soil and groundwater from historical land uses, comprehensive air emissions quality modelling, noise modelling, human health, transportation, and waste management. Environmental studies will inform design mitigations of environmental effects and the basis for refinery-specific management plans.

24.2.10.2 Permitting

Permitting of a metals refinery in BC is subject to existing well-defined processes. The BC *Environmental Assessment Ac*t Reviewable Projects Regulation (BC. Reg. 32/2023), specifies thresholds for the evaluation of a "metals refining" facility under Part 2 – Industrial Projects, Table 2, Primary Metals Industry. If the Baptiste Refinery Option was integrated with mine operations, and in the vicinity of the mine site and dedicated to the mining operations at the mine site would be deemed not reviewable through a separate Environmental Assessment as per Column 2, criteria 2) of the Regulation, and in this case the Refinery Option provincial environmental assessment would occur as a facility scoped into the Baptiste Project EA (see section 20.2).

Federal government assessment would be determined by whether the Refinery Option triggered federal thresholds, for example under the *Fisheries Act*, as the Canada *Impact Assessment Act* Physical Activities Regulation (SOR/2019-285) Schedule 2 listing designated projects does not identify a metals refinery as a designated project under the IAAC. In addition to receipt of an EA Certificate approving the refinery, the Refinery Option will require provincial and local government permits to construct, operate and ultimately close the refinery operations. Permits for discharge of waste to the environment (air and effluent discharges) will require detailed design and modelling. Municipal permitting may require re-zoning of land use if existing land use bylaws do not presently list a refinery as a listed industrial use, development permits and permits to connect to municipal utilities.

24.2.10.3 Social and Community

Local perspectives and early engagement are critical to FPX's decision-making process throughout all aspects of the Project. Enduring relationships must be built with Indigenous rightsholders, based on transparency, accountability, mutual understanding, and respect for Indigenous rights and title, and a long-term commitment to shared value. Enduring relationships will also need to be built with local government, residents, business and service providers in the community where the Refinery Option will be sited. It is anticipated that the EA will establish the scope of studies and engagement required to complete the social and economic assessment of the Refinery Option.

If located in central BC, the Refinery Option site will include Indigenous communities and traditional territories which are in addition to those which have been identified for the Baptiste Project in Section 20.3.

24.2.10.4 Closure and Reclamation

It is envisioned that following the end of operations, the refinery will be decommissioned and any remaining dry waste tailings will be disposed of in the tailings facility at the Baptiste site. Any remaining aluminum and chromium waste will be properly disposed of at appropriate hazardous waste disposal facilities.

24.2.11 Capital and Operating Costs

24.2.11.1 Introduction

The capital and operating cost estimates presented in this section provide substantiated costs that can be used to assess the preliminary economics of the Refinery Option. The estimates are based on the construction of a refinery, oxygen plant, and sulphuric acid plant, as well as owner's costs and provisions.

24.2.11.2 Capital Costs

24.2.11.2.1 Capital Costs Summary

The capital cost estimate conforms to Class 5 guidelines of AACE International, with an estimated accuracy of +50%/-35%. The capital cost estimate was developed in Q3 2023 US$ and is based on a combination of Ausenco's in-house database of projects and advanced studies, budgetary quotes for specialized equipment, and experience from similar operations.

The total initial capital cost for the Refinery Option is US$448 M. The capital cost summary is presented below in Table 24-16. Table 24-17 provides a summary of the same capital costs for the Refinery Option, categorized by the work breakdown structure (WBS) of the capital cost estimate.

A contingency of 30% of the combined indirects and directs was applied.

Table 24-16: Capital Cost Summary – Refinery Option

Capital Category	Initial Capital (US$M)
Hydrometallurgical Refinery	
Leaching – Feed Preparation	1.8
Leaching – Pressure Leach	31.3
Leaching – Atmospheric Leach	7.0
Oxygen Plant	45.8
Cobalt Solvent – Extraction and Recovery	14.1
Nickel Solvent – Extraction	22.5
Nickel Sulphate – Crystallization, Drying, Packaging	33.3
Sodium Sulphate Rejection	42.2
Reagents – Storage and Distribution	3.4
Subtotal	**201.4**
On-site Infrastructure	
Infrastructure Buildings (Office, Maintenance Shop, Warehouse)	10.0
Site Water Management	2.0
Site Services	3.0
Sulphuric Acid Plant	40.0
Subtotal	**55.1**
Off-site Infrastructure	
Main Access Road	0.3
Water Supply	0.5
Power Supply	4.3
Subtotal	**5.1**
Total Direct Costs	**261.5**
Total Indirect Costs	72.9
Owner's Costs	13.0
Contingency	100.4
Total Capital Cost	**447.8**

Table 24-17: Capital Cost Summary, by WBS Code – Refinery Option

WBS	Description	Total Cost (US$M)
6110	Leaching – Feed Preparation	1.8
6120	Leaching – Pressure Leach	31.3
6130	Leaching – Atmospheric Leach	7.0
6140	Oxygen Plant	45.8
6150	Cobalt Solvent – Extraction and Recovery	14.1
6160	Nickel Solvent – Extraction	22.5
6170	Nickel Sulphate - Crystallization, Drying, Packaging	33.3
6180	Sodium Sulphate Rejection	42.2
6190	Reagents – Storage and Distribution	3.4
	Total Hydrometallurgical Refinery	**201.4**
6230	Infrastructure Buildings (Office, Maintenance, Shop, Warehouse)	10.0
6240	Site Water Management	2.0
6250	Site Services	3.1
6260	Sulphuric Acid Plant	40.0
	Total On-site Infrastructure	**55.1**
6310	Main Access Road	0.3
6320	Water Supply	0.5
6330	Power Supply	4.3
	Total Off-site Infrastructure	**5.0**
	Total Direct Costs	**261.5**
6410	Field Indirects	15.7
6420	Project Delivery	44.4
6430	Commissioning Operations Readiness	0.8
6440	Vendor Representatives	3.9
6450	Spares	4.1
6460	First Fills	3.9
6470	Freight Costs (With Direct Costs)	-
	Total Indirect Costs	**72.9**
6480	Owner's Costs	13.0
	Total Owner's Costs	**13.1**
6490	Contingency	100.4
	Total Contingency Costs	**100.4**
	Total Capital Costs	**447.8**

24.2.11.2.2 Basis of Capital Cost Estimate

The basis for which the capital cost estimate was conducted is as follows:

- conceptual engineering design by Ausenco;
- characteristics of the concentrate from the proposed Baptiste concentrator, as described in the other sections in this report;
- bulk factoring for earthworks, concrete, structural steel, platework, piping, and electrical on the total installed mechanical equipment cost for each circuit;
- field indirects factored at 6% of total direct costs;
- project delivery costs factored at 17% of total direct costs;
- commissioning factored at 1% of total equipment supply;
- vendor representatives factored at 1.5% of total direct costs;
- spares (commissioning, critical, and operating) factored at 5% of total equipment supply;
- first fills factored at 1.5% of total direct costs;
- owner’s costs factored at 5% of total direct costs;
- costs escalated to Q3 2023 when using historical pricing;
- sustaining capital costs not considered at this level of estimate; and
- contingency allowance of 30% of total direct and indirect costs included.

24.2.11.2.3 Area 6100 – Refinery Capital Cost

This WBS area covers all pieces of equipment directly pertaining to the operation of the refinery, such as items for the oxygen plant, feed preparation, leaching, solvent extraction, impurities removal, and reagent storage, and distribution.

24.2.11.2.4 Area 6200 – On-site Infrastructure

The on-site infrastructure is grouped into the following main categories:

- bulk earthworks;
- on-site power supply and distribution;
- infrastructure buildings;
- site water management; and
- site services.

The list of buildings falling under WBS 6230 and 6260 can be found in Table 24-14.

On-site power supply and distribution includes all transformers, switchgear, motor control centers (MCCs), standby generator, and on-site powerlines. The off-site substation cost is included in off-site infrastructure (see Section

24.1.11.7). Site water management includes the distribution of water to consumers in the refinery, wastewater management, and surface water control. Site facilities include fuel storage and waste disposal.

24.2.11.2.5 Area 6300 – Off-site Infrastructure

The off-site infrastructure is grouped into the following main categories:

- main access road;
- water supply; and
- power supply.

This estimate allows for the connection of the main access road to city infrastructure and for the connection to the municipal water supply.

The cost of connection to BC Hydro's off-site power system is described in Section 24.1.8.3.

24.2.11.2.6 Area 6400 – Indirect Costs

The following details the breakdown of the indirect costs for the refinery. Note that freight costs are included in the direct cost estimate.

- field indirects;
- project delivery;
- commissioning;
- vendor representatives;
- spares;
- first fills; and
- freight.

24.2.11.2.7 Closure and Reclamation

A closure and reclamation cost estimate has not been prepared for this phase of study.

24.2.11.3 Operating costs

24.2.11.3.1 Introduction

The operating cost for the refinery is based on a 40 kt/a nickel in nickel sulphate production and increased oxygen and sulphuric plant capacity to produce sulphuric acid for both the refinery and main concentrator at Baptiste. The annual estimated operating cost for the refinery will be US$98.1M or US$2,449 /t nickel produced Table 24-18. With the credit for sulphuric acid sales to the main concentrator included, the net operating cost for the refinery will be US$90.0M, or US$2,244 /t nickel produced, which equates to US$1.02/lb nickel.

Table 24-18: Operating Cost Summary (Acid Credits Not Included)

Operating Cost Summary (No H2SO4 Credits)	Total (US$M/a)	US$/t Ni
Power	5.6	139.4
Reagents and Consumables	64.8	1,618.2
Labour	12.9	321.5
Maintenance	2.9	71.4
Product Transportation	7.0	175.0
G&A	4.9	123.2
Total	**98.1**	**2,449**

Values may not sum due to rounding.

The following subsections detail the breakdown of different operating cost centers.

24.2.11.3.2 Power

The operating power consumption is presented in Table 24-19.

The largest cost driver in terms of power consumption will be the reagents and utilities areas. The primary contributors to the power consumption in these areas are the sulphuric acid and oxygen plants, which will have an operating consumption of 2.4 GWh/a and 47.5 GWh/a, respectively.

The primary power consumers in the nickel solvent extraction and magnesium and sodium removal circuits are their respective crystallizers. The nickel crystallizer package, including centrifuge and steam compressor, is 15.5 GWh/a. For the sodium crystallizer package, the total annual power consumption will be 30.8 GWh/a. The basis for power cost is the same as the concentrator, using the rate provided by the provincial utilities provider.

Table 24-19: Power Cost Breakdown

Area	Operating Consumption (GWh/a)	Cost ($M/a)
Leaching	4.1	0.2
Cobal Solvent Extraction and Recovery	1.0	0.05
Nickel Solvent Extraction and Recovery	19,4	1.0
Magnesium and Sodium Removal	34.8	1.7
Reagents and Utilities	52.4	2.6
Total	**111.8**	**5.6**

24.2.11.3.3 Reagents and Consumables

The reagent and consumable operating cost breakdown are presented in Table 24-20.

Sodium hydroxide consumption is the largest cost driver by a large margin; the majority of consumption in the NiSX process.

Molten sulphur will be consumed in the sulphuric acid plant, with the plant producing enough acid to supply both the refinery and concentrator.

Table 24-20: Reagents and Consumables Cost Breakdown

Reagent and Consumable	Cost (US$M/a)
Molten Sulphur	4.9
Sodium Hydroxide (NaOH)	58.1
Flocculant	0.02
Extractant	0.2
Diluent	0.07
Product Bags	1.1
Fresh Water	0.08
Steam	0.2
Filter Consumables	0.2
Total	**64.8**

24.2.11.3.4 Labour

The labour force will primarily be composed of local labourers residing in Prince George or nearby communities.

The labour rate and salary burden are consistent with the concentrator labour force described in Section 17 for similar roles. Table 24-21 summarizes the annual cost of the total workforce.

Table 24-21: Labour Cost Breakdown – Total Workforce

Area	No. of Personnel	Cost (US$M/a)
Management	32	2.6
Technical Services	18	1.7
Maintenance	18	1.7
Leach	16	1.3
Cobalt Solvent Extraction and Recovery	16	1.3
Nickel Solvent Extraction and Recovery	16	1.3
Magnesium and Sodium Removal	16	1.3
Laboratory	20	1.5
Product Transportation	4	0.3
Total	**156**	**12.9**

24.2.11.3.5 Maintenance

Maintenance consumables were estimated by applying a factor to the total mechanical equipment cost for each specific area (see Table 24-22). A 5% factor was applied to the leaching for the leaching circuit, which is consistent with Ausenco's experience in autoclave leach circuits and industry benchmarks. For the remaining process areas, a 3% factor was applied.

Table 24-22: Maintenance Consumables Cost Breakdown

Area	Cost (US$M/a)
Leaching	0.6
Cobalt Solvent Extraction and Recovery	0.1
Nickel Solvent Extraction and Recovery	0.5
Magnesium and Sodium Removal	0.4
Reagents and Utilities	1.2
Total	**2.8**

24.2.11.3.6 Product Transportation

Nickel sulphate, cobalt hydroxide, and copper cement will be packaged in lined bags and loaded into containers for truck transport to the nearest rail. These three products will then be transported by rail for distribution to contract holders.

Sodium sulphate will be bagged and free-issued to regional consumers, primarily kraft pulp mills. In consideration of this free issue, any cost of transport and revenue from the sales of sodium sulphate are assumed to break even, therefore the transport cost of sodium sulphate is not included in the operating cost estimate.

Table 24-23: Product Transportation Cost Breakdown

Product	Cost (US$M/a)
Nickel Sulphate	6.9
Cobalt Hydroxide	0.06
Copper Cement	0.02
Sodium Sulphate	-
Total	**7.0**

24.2.11.3.7 General and Administration (G&A)

G&A costs are expenses not directly related to the production of the refinery concentrate and are expenses that are not covered under processing, labour, and transportation costs. The G&A costs were developed using a combination of Ausenco's in-house data on existing operations and publicly available benchmark data. Table 24-24 summarizes G&A by category.

Table 24-24: G&A Cost Breakdown

Description	Cost (US$M/a)
General Administration Costs	1.6
Personnel Transportation	0.1
Catering and Housekeeping	1.3
Laboratory Costs	1.0
Insurance	1.0
Waste Disposal	0.03
Total	**4.9**

24.2.12 Economic Analysis

24.2.12.1 Introduction

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to both known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes:

- Mineral Resource and reserve estimates.
- Assumed commodity prices and exchange rates.
- The proposed mine production plan.
- Projected mining and metallurgical recovery rates.
- Sustaining costs and proposed operating costs.
- Assumptions as to closure costs and closure requirements.
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what are estimated.
- Unrecognized environmental risks.
- Unanticipated reclamation expenses.
- Unexpected variations in quantity of mineralized material, grade, or recovery rates.
- Geotechnical or hydrogeological considerations during mining being different from what was assumed.
- Failure of mining methods to operate as anticipated.
- Failure of plant, equipment, or processes to operate as anticipated.
- Changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis.
- Ability to maintain the social licence to operate.
- Accidents, labour disputes and other risks of the mining industry.
- Changes to interest rates.
- Changes to tax rates.
- Deviations in commodity prices from those assumed in the financial model.

The information provided in this section relates to the economic analysis of the project's Refinery Option, which is separate from the PFS Base Case. Section 22 contains the economic analysis of the PFS Base Case, without any consideration for the Refinery Option.

24.2.12.2 Methodologies Used

As part of the Refinery Option analysis, two scenarios were considered:

- The PFS Base Case + Refinery Option scenario.
- The Refinery Option Only scenario.

An engineering economic model was developed to estimate annual pre-tax and post-tax cash flows and economics for each scenario, based on an 8% discount rate. Costs and financials for the Refinery Option Only are derived by simplistic delta cashflow analysis and not modeled as a standalone project. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations. For the PFS Base Case + Refinery Option scenario, a post-tax sensitivity analysis was performed to assess impact of variations in discount rate, nickel price, metallurgical recovery, operating costs, and initial capital costs. The capital and operating cost estimates were developed specifically for both scenarios and are summarized in Section 21 and Section 24.1.11 of this report (presented in Q3 2023 dollars). It is important to note that the refinery estimates were conducted at a lower level of accuracy compared to the PFS Base Case and is used to evaluate the strategic potential of further refining the PFS Base Case concentrates. Similar to the Base Case, the economic analysis has been run with no inflation (constant dollar basis).

24.2.12.3 Financial model Parameters

The economic analysis of both scenarios was performed using the following assumptions:

- construction period of 36 months;
- mine life of 29 years;
- a nickel price of US$8.75/lb with an additional US$1.00/lb premium for nickel sulphate sold;
- cobalt price of US$15.00/lb;
- copper price of US$3.50/lb;
- United States to Canadian dollar exchange rate assumption of C$1.00 : US$0.76;
- cost estimates in constant 2023 US$ with no inflation or escalation factors considered;
- no salvage value assumed;
- results are based on 100% ownership, with:
  - a 1.0% NSR considered for the PFS Base Case + Refinery Scenario; and
  - no NSR considered for the Refinery Option Only Scenario.
- capital costs funded with 100% equity (i.e., no financing costs assumed);
- all cash flows discounted to beginning of construction;
- all metal products are assumed sold in the same year they are produced; and
- project revenue is derived from the sale of awaruite concentrate briquette, nickel sulphate, cobalt, and copper.

24.2.12.4 Taxes

The economic analysis of both scenarios uses the same tax assumptions as detailed in Section 22.3.1 of this report.

24.2.12.5 Working Capital

Working capital assumptions include accounts receivable (0 days), inventories (30 days), and accounts payable (30 days). The effective sum of working capital over the life of mine is zero.

24.2.12.6 Economic Analysis

The economic analysis was performed assuming an 8% discount rate, with a summary of the economics for both scenarios included in Table 24-25.

Table 24-25: Summary of Project LOM Cashflow Assumptions and Results – Refinery Option

General	Unit	PFS Base Case + Refinery Option	Refinery Option Only***
		LOM Total/Avg.	
Nickel Price	US$/lb	8.75	8.75
Nickel Sulphate Premium (in addition to Nickel Price)	US$/lb	1.00	1.00
Cobalt Price	US$/lb	15.00	15.00
Copper Price	US$/lb	3.50	3.50
Exchange Rate	C$:US$	0.76	0.76
Discount Rate	%	8	8
Mine Life	years	29	29
Production	**Unit**	**LOM Total/Avg.**	
Nickel Produced - Awaruite Concentrate	mlbs	1,263	--
Nickel Produced - Nickel Sulphate	mlbs	2,499	2,499
Total Nickel Produced	mlbs	3,761	2,499
Total Average Annual Nickel Production	kt/a	58.8	39.1
Nickel Payability - Awaruite Concentrate	%	95	--
Nickel Payability - Nickel Sulphate	%	100	100
Total Payable Nickel	mlbs	3,698	2,499
Total Cobalt Produced	mlbs	40	40
Total Average Annual Cobalt Production	kt/a	0.6	0.6
Cobalt Payability	%	87	87
Total Payable Cobalt	mlbs	35	35
Total Copper Produced	mlbs	19	19
Total Average Annual Copper Production	kt/a	0.3	0.3
Copper Payability	%	95	95
Total Payable Copper	mlbs	18	18
Operating Costs	**Unit**	**LOM Total/Avg.**	
Mining Cost	US$/t Milled	3.14	--
Processing Cost	US$/t Milled	3.63	--
G&A Cost	US$/t Milled	1.09	--
Refinery Cost	US$/t Milled	1.71	1.71
Transportation Cost	US$/t Milled	0.29	--
Total Operating Cost	US$/t Milled	9.87	1.71
Transportation and Royalties	**Unit**	**LOM Total/Avg.**	
Royalty NSR	%	1	0
C1 and AISC	**Unit**	**LOM Total/Avg.**	
C1 Cost*	US$/lb Ni	3.89	0.79
All-in Sustaining Cost (AISC)**	US$/lb Ni	4.38	0.79
Capital Costs	**Unit**	**LOM Total/Avg.**	
Initial Capital (Including Refinery)	US$M	2,629	448
Sustaining Capital	US$M	1,181	--
Growth Capital	US$M	763	--
Closure Costs	US$M	284	--
Financials - Pre-Tax	**Unit**	**LOM Total/Avg.**	
Pre-Tax NPV (8%)	US$M	3,046	78
Pre-Tax IRR	%	17.9	9.9
Pre-Tax Payback	years	5.0	8.7
Financials - Post-Tax	**Unit**	**LOM Total/Avg.**	
Post-Tax NPV (8%)	US$M	2,127	63
Post-Tax IRR	%	17.7	9.9
Post-Tax Payback	years	3.9	7.5

* C1 costs consist of operating cash costs plus treatment and refining charges for produces excluding Nickel Sulphate, and transportation cost.
** AISC consist of total cash costs plus royalties, sustaining capital, BC hydro integration, and closure cost.
*** Costs and financials for the Refinery Option Only are derived by simplistic delta cashflow analysis and not modeled as a standalone project.

24.2.12.7 Sensitivity Analysis

For the PFS Base Case + Refinery Option Scenario, a post-tax sensitivity analysis was performed to assess impact of variations in discount rate, nickel price, concentrator metallurgical recovery, operating costs, and initial capital costs. Post-tax sensitivity analysis results are summarized in Table 24-26, Figure 24-6 and Figure 24-7. Analysis revealed that the project's NPV and IRR are most sensitive to changes in nickel price and concentrator metallurgical recovery, and due to the 29-year mine life, the project's NPV is more sensitive to operating cost than initial capital cost.

Table 24-26: Post-Tax Sensitivity Analysis Results – PFS Base Case + Refinery Option Scenario

Post-Tax NPV (US$M) Sensitivity To Discount Rate

Discount Rate	Commodity Price					
		(20%)	(10%)	0%	10%	20%
	5%	2,009	2,934	3,855	4,774	5,693
	8%	880	1,507	2,127	2,744	3,360
	10%	403	902	1,393	1,882	2,368

Post-Tax IRR % Sensitivity To Discount Rate

Discount Rate	Metal Price					
		(20%)	(10%)	0%	10%	20%
	5%	12.4%	15.2%	17.7%	19.9%	22.1%
	8%	12.4%	15.2%	17.7%	19.9%	22.1%
	10%	12.4%	15.2%	17.7%	19.9%	22.1%

Post-Tax NPV (US$M) Sensitivity To Initial Capital Cost

Initial Capital	Metal Price					
		(20%)	(10%)	0%	10%	20%
	(20%)	1,140	1,759	2,376	2,991	3,604
	(10%)	1,010	1,634	2,251	2,868	3,482
	--	880	1,507	2,127	2,744	3,360
	10%	748	1,378	2,001	2,619	3,236
	20%	614	1,249	1,874	2,494	3,112

Post-Tax IRR Sensitivity To Initial Capital Cost

Initial Capital	Metal Price					
		(20%)	(10%)	0%	10%	20%
	(20%)	14.8%	17.9%	20.7%	23.3%	25.7%
	(10%)	13.5%	16.4%	19.1%	21.5%	23.7%
	--	12.4%	15.2%	17.7%	19.9%	22.1%
	10%	11.5%	14.1%	16.5%	18.6%	20.6%
	20%	10.7%	13.2%	15.4%	17.5%	19.4%

Post-Tax NPV (US$M) Sensitivity To Operating Cost

Operating Cost	Metal Price					
		(20%)	(10%)	0%	10%	20%
	(20%)	1,427	2,048	2,665	3,281	3,895
	(10%)	1,154	1,778	2,396	3,013	3,628
	--	880	1,507	2,127	2,744	3,360
	10%	604	1,235	1,857	2,475	3,092
	20%	324	961	1,587	2,205	2,823

Post-Tax IRR Sensitivity To Operating Cost

Operating Cost	Metal Price					
		(20%)	(10%)	0%	10%	20%
	(20%)	14.8%	17.3%	19.6%	21.8%	23.8%
	(10%)	13.7%	16.3%	18.7%	20.9%	23.0%
	--	12.4%	15.2%	17.7%	19.9%	22.1%
	10%	11.1%	14.0%	16.6%	19.0%	21.2%
	20%	9.7%	12.8%	15.5%	18.0%	20.2%

Post-Tax NPV (US$M) Sensitivity To Recovery

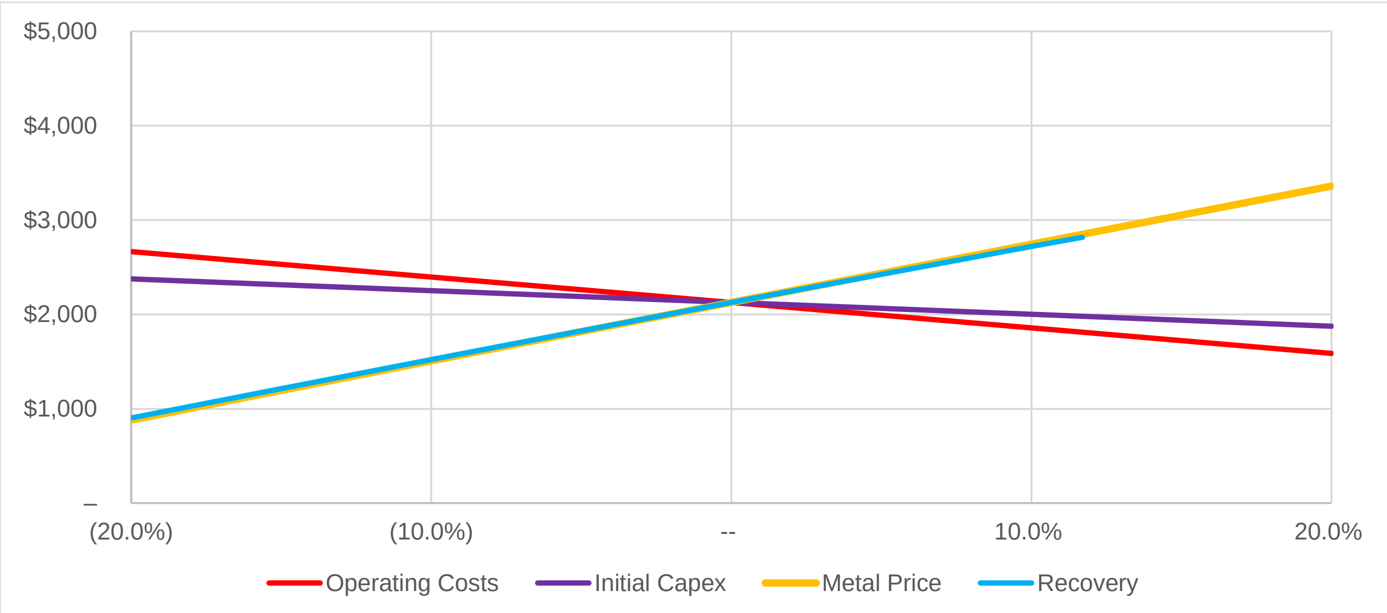
Recovery	Metal Price					
		(20%)	(10%)	0%	10%	20%
	(20%)	(134)	390	902	1,407	1,907
	(10%)	382	956	1,521	2,080	2,638
	--	880	1,507	2,127	2,744	3,360
	10%	1,363	2,043	2,720	3,395	4,068
	20%	N/A	N/A	N/A	N/A	N/A

Post-Tax IRR Sensitivity To Recovery

Recovery	Metal Price					
		(20%)	(10%)	0%	10%	20%
	(20%)	7.2%	10.1%	12.5%	14.7%	16.8%
	(10%)	10.0%	12.8%	15.2%	17.5%	19.5%
	--	12.4%	15.2%	17.7%	19.9%	22.1%
	10%	14.6%	17.3%	19.9%	22.2%	24.4%
	20%	N/A	N/A	N/A	N/A	N/A

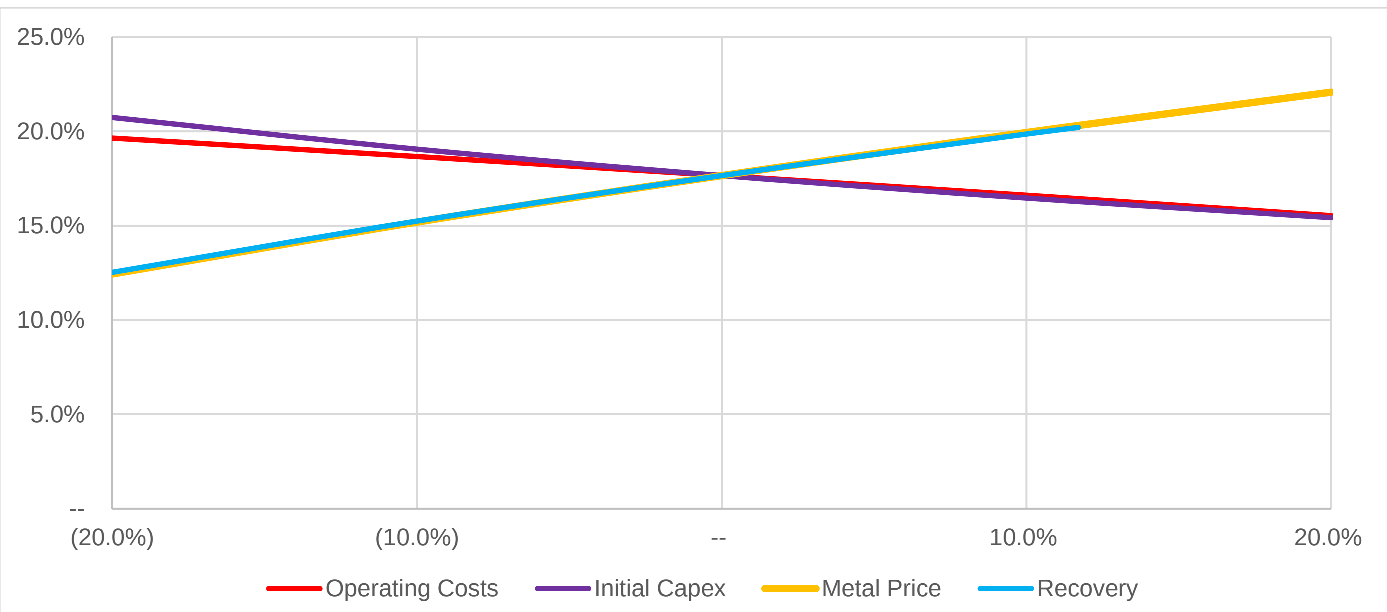
Source: Ausenco, 2023.

Figure 24-6: PFS Base Case + Refinery Option Scenario: Post-Tax NPV 8% Sensitivity Chart (US$M)



Source: Ausenco, 2023.

Figure 24-7: PFS Base Case + Refinery Option Scenario: Post-Tax IRR Sensitivity Chart



Source: Ausenco, 2023.
Notes:
- In Figure 24-6 and Figure 24-7 chart lines for nickel price and metallurgical recovery overlap.
- In Figure 24-6 and Figure 24-7 chart lines for metallurgical recovery end as recovery reaches 100%

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs have provided the following interpretations and conclusions in their respective areas of expertise based on the review of data available for this report.

25.2 Mineral Tenure, Surface Rights, Water Rights, Royalties

The Decar Property comprises 62 contiguous mineral claims that cover 24,740 hectares (247 km^2) in the Omineca Mining Division of central BC, Canada. The claims confer title to subsurface mineral tenure only, as defined by the Mineral Tenure Act of BC.

All claims are registered in the name of, and are 100% owned by, FPX. There is a 1% NSR royalty over the Decar Project owned by a Lender who secured the repurchase of the 60% interest in the Property that had been acquired by Cliffs Natural Resources Ltd.

Surface rights over the Decar Property are mostly owned by the Crown and administered by the Government of BC and would be available for any eventual mining operation. Parts of several claims overlap with District Lots, Mineral Reserve Sites, and Rubyrock Lake Provincial Park and so are either off limits or potentially off limits to mineral exploration and development. However, these overlaps comprise just 3% of the total area of the Decar Property and occur at least 2.5 km from known awaruite mineralization, so they are not considered material to the project.

25.3 Geology and Mineralization

Cache Creek terrane that underlies the Decar Property is subdivided into (from southwest to northeast) the Rubyrock igneous complex, Sitlika assemblage, Trembleur ultramafite, and Sowchea succession. These subdivisions are structurally intercalated by faults internal to the Cache Creek terrane. Awaruite mineralization occurs entirely within the Trembleur ultramafite and is correlated to increased serpentinization.

Historical exploration on the Property has focused on three deposit types: (1) awaruite in serpentinized ultramafic rock, (2) listwanite-hosted gold, and (3) chromite pods in ultramafic. At time of report, only the awaruite deposits are potentially economic.

25.4 Exploration, Drilling, and Analytical Data Collection in Support of Resource Estimation

To date, completed exploration programs show that rock sampling of outcrop and magnetic surveys are effective methods for identifying potential awaruite mineralization on the Decar Property.

FPX has completed a total of 131 holes for 40,652 m of diamond drilling, with approximately 80-85% of holes and metres drilled on the Baptiste Deposit. Awaruite mineralization is visible in drill core and correlates with high magnetic susceptibility and strong serpentinization.

Sampling methods are acceptable and well-monitored to support mineral resource estimation. Sample preparation, analysis, and security were performed in accordance with exploration best practices and industry standards at the time the information was collected.

The quality and quantity of the geological data, collar, and downhole survey data collected in the exploration and infill drill programs are sufficient to support mineral resource estimation.

No material factors were identified with the data collection from the drill programs that could significantly affect mineral resource estimation.

Sample preparation, and analyses were performed by independent accredited laboratories. The sample preparation, analysis, and security practices are acceptable, meet industry-standard practices at the time they were undertaken, and are sufficient to support mineral resource estimation.

The data verification programs concluded that the data collected from the project adequately support the geological interpretations and that the database is of sufficient quality to support the use of the data in mineral resource estimation.

25.5 Metallurgical Testwork

Metallurgical test work to support the evaluation of the Baptiste Deposit has been completed under the supervision of the QP during the last seven years, at a number of reputable Canadian metallurgical research facilities. Samples for use in metallurgical test work were identified by working with FPX geological staff familiar with the Baptiste Deposit. The number of metallurgical samples in the various test work programs, and the size of these samples, has also ensured that the deposit has been representatively sampled for the purpose of conducting metallurgical studies. It is the opinion of the QP that the metallurgical performance of the Baptiste Deposit is well understood at this time.

25.6 Mineral Resource Estimate

The QP believes that the mineral resource estimates for the Baptiste Deposit have been generated using industry-standard methods that follow procedures recommended by the CIM in the “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” (CIM, 2019). As such, the resource block model and its global resource inventory are suitable for public disclosure and for further use in the preliminary feasibility of the technical and economic viability of the project.

25.7 Mining

25.7.1 Geotechnical Considerations

Pit slope angles vary by sector within the open pit and are based on recommendations by Knight Piésold (Preliminary Open Pit Slope Assessment, dated November 1, 2021). The resulting pit slope designs are based on double-benching of 10 m high benches and consider haulage ramp positioning, safety berms and other geotechnical features required to maintain safe inter-ramp and overall slope angles.

Slope recommendations are summarized into three categories, namely overburden, south wall, and north wall. The respective pit slope design parameters that have been used are presented in Table 25-1.

Table 25-1: Mine Design Pit Slope Recommendations

Design Sector	Max Wall Height (m)	Bench Height (m)	Bench Face Angle (°)	Bench Width (m)	Inter-ramp Angle (°)	Max. Inter-ramp Slope Height (m)	Max. Overall Slope Angle (°)
Overburden	30	10	40	8.0	27	30	
South Wall Bedrock	500	20	70	9.0	51	150	44
North Wall Bedrock	700	20	65	8.6	48	150	42

25.7.2 Mine Plan

The Baptiste Deposit will be mined via bulk open-pit mining operations that will be developed in two phases. Phase 1 includes the first 9 years of operation (plus the preceding 2 years of mine pre-stripping) with a nominal ore processing rate of 108 kt/d. Phase 2 begins in Operating Year 10, is 19 years in duration, and coincides with a plant expansion to a nominal ore processing rate of 162 kt/d.

The Baptiste Deposit will be mined as a conventional large-scale truck and shovel operation with up to 60 Mt of material mined per annum during Phase 1, and up to 120 Mt of material mined per annum during Phase 2. The mining operation will feature 250 mm blast hole electric drills, 42 m^3 electric excavators, and 300 t class haul trucks working on nominal 10 m high benches. A flexible combination of dozers, graders, wheel loaders, and excavators will form the core of the support equipment fleet.

Over the project's 29-year life, the concentrator will be fed with ore directly from the mine, and in the final year from the operational ore stockpile. The production of the first year is 29.6 Mt (82 kt/d), which represents 75% of the concentrator's nameplate capacity. Throughput is restricted this year as the plant commissioning is completed and throughput is ramped-up. In Years 2 through 9, the full 39.4 Mt (108 kt/d) of ore are delivered to the primary crusher. During this period, the mine ramps-up from approximately 15 Mt in Year -2 up to 50 Mt in Year 3. Then, from Year 4 through 9 the mine moves an average of 60 Mt per annum. In Year 10, a third grinding line will be commissioned and the average annual concentrator throughput will increase to 59.1 Mt per annum (162 kt/d). During Years 10 through 29, the mine reaches a peak mining capacity of 120 Mt per annum, with an average mine capacity of 96 Mt per annum.

25.8 Recovery Methods

The recovery methods and equipment align with conventional practices in the industry. Ore comminution, magnetic separation, flotation, leaching, and precipitation recovery of payable metals and handling of tailings are all commonly used in industry. Previous studies, coupled with new testwork results, were used to develop the resulting flowsheet suitable for each stage over the LOM.

The process plant has been designed for nickel recovery to produce a main product awaruite concentrate by magnetic separation and flotation treatment, and a MHP coproduct by precipitation of the nickel leached into solution in flotation and flotation tailings leach circuits. The plant will be constructed in two phases. The initial plant capacity is 108 kt/d, followed by an expansion to 162 kt/d for Year 10 operations. The Phase 2 expansion is conservatively considered as a standalone concentrator in this study, and this approach could potentially be optimized in future studies.

25.9 Environmental, Permitting and Social Considerations

Cultural and environmental baseline studies re-commenced in 2021 and the Baptiste PFS has benefitted from learnings from 18 months of baseline data collection, as well as from historic studies completed from 2011-2014. FPX has also initiated early engagement with local communities and rightsholders, which will continue to inform the Project design and baseline study scoping. Key learnings from baseline studies to date include for example, the very low likelihood for potentially acid generating tailings or waste rock materials, and early identification of species and plants of special interest in the Project area. All of the information being gathered in the Project area, as well as ongoing early engagement initiatives, will contribute to future engineering studies and development of the Initial Project Description to initiate the EA process.

25.10 Capital Cost Estimate

The initial, expansion, and sustaining capital cost estimate conforms to Class 4 guidelines for a PFS-level estimate with a ±25% accuracy according to AACE International. The capital cost estimate was developed in Q3 2023 US$ based on budgetary quotations for equipment as well as Ausenco's in-house database of projects and studies, including experience from similar operations.

25.11 Operating Cost Estimate

The operating cost estimate was developed in Q3 2023 US$ from budgetary quotations and Ausenco's in-house database of projects and studies as well as experience from similar operations. Mine operating costs have been estimated from base principles using quotations from local mine equipment vendors plus local supply consumables. The accuracy of the operating cost estimate is ±25%. The estimate includes mining, processing, G&A costs, and product transportation.

25.12 Economic Analysis

The economic analysis was performed assuming an 8% discount rate. The pre-tax NPV discounted at 8% is US$2,923 M, the IRR is 19.1%, and payback is 4.6 years. On an after-tax basis, the $NPV_{8\%}$ is US$2,010 M, the IRR is 18.6%, and the payback period is 3.7 years. The after-tax payback period is shorter than the pre-tax payback period due to the inclusion of investment tax credit refunds in Year -1 to Year 2 of the Project.

A sensitivity analysis was conducted on the base case pre-tax and post-tax 8% NPV and IRR of the Project, using the following variables: discount rate, nickel price, metallurgical recovery, operating costs, and initial capital costs. Analysis revealed that the Project is most sensitive to changes in nickel price and metallurgical recovery. Reflective of the 29-year mine life, the Project is more sensitive to operating costs than initial capital.

The prefeasibility study supports a decision to carry out additional detailed studies.

25.13 Refinery Option

A 40 kt/a nickel refinery has been designed to explore an opportunity where a portion of the awaruite concentrate and all the MHP from the Baptiste concentrator can be further refined into battery grade nickel sulphate, cobalt-rich MHP, and copper cement. This Refinery Option demonstrates good economic returns of the refinery whilst validating the strategic flexibility of awaruite concentrate, allowing for integration to both the stainless steel and battery material supply chains.

25.14 Risk and Opportunities

25.14.1 Risks

25.14.1.1 Exploration

The operational risks for the exploration program at Baptiste are the same as those experienced by any exploration program: permitting, access, mineral title, and personal security. Provided FPX continues to meet the obligations of its surface access agreements, applicable regulations, and existing exploration permits, no operational difficulties are anticipated. Review of project data did not identify any significant risks or uncertainties that could be reasonably expected to affect the reliability or confidence in the exploration information summarized in this report.

25.14.1.2 Metallurgy

The risk to the implementation of the proposed metallurgical process plan, as outlined, is low. There is a low percentage risk that sampling of the deposit used in the defining metallurgical test work has not properly characterized the realized mine production. This risk is associated with every greenfield project and is thought to be minimized by the observed uniform nature of the Baptiste Deposit. There is little to no risk that the metallurgical test work has been poorly performed as numerous laboratory facilities have participated in the completion of the test work development in various phases. The fact that the process does not have an industry example to follow and is considered unique could be considered a risk, although the unit operations employed have long standing acceptance within industry in different applications.

25.14.1.3 Processing

The main risks identified for the Baptiste processing circuit are listed below.

25.14.1.3.1 Primary Crusher Feed Coarser than Target

The proposed primary crusher is selected based on a fine ROM ore size distribution achieved through intensive blasting practices. However, if a coarser ROM ore size is delivered to the primary crusher, it could cause production loss or impact metal recovery due to a slightly coarser primary grind size, which may impact primary magnetic separation recovery and regrind requirements. However, due to the intense magnetic response of awaruite, this risk is somewhat mitigated.

25.14.1.3.2 Gearless Mill Drives (GMD) of SAG Mills

GMD systems are required for the two selected SAG mills. A GMD system comes with a higher capital cost and GMD costs have been volatile over the last two years. It also involves a longer time for equipment supply, installation, and commissioning. These could bring potential cost and schedule risks for the project.

25.14.1.3.3 SAG Mills Discharge Handling

The SAG mill discharge system involves a trommel only discharge arrangement with no pebble crushing stage considering an estimated low pebble production rate. If pebble production rates exceed design, however, potential operational risks associated with the arrangement would arise. Further investigations of the hardness and competency variability of the mineralized material is recommended to fully understand and mitigate the risk.

25.14.1.3.4 Awaruite Tailings Leach

The tailings leach circuit design is based on limited test work that could be further optimized in terms of acid consumption and circuit performance. The potential risk includes a higher process operating cost and constrained nickel recovery projection. Additional testwork will mitigate the risk.

25.14.1.3.5 Brine Purification and Precipitation

The purification and precipitation circuit design is based on typical industry benchmark data. This needs to be verified to avoid negative impacts on flowsheet development, equipment sizing, and performance projection. Additional testwork will mitigate the risk.

25.14.1.4 Mineral Resource Estimate

Areas of uncertainty that may materially impact the mineral resource estimate include:

- commodity price assumptions;
- pit slope angles;
- metal recovery assumptions; and
- mining and process cost assumptions.

There are no other known factors or issues that materially affect the estimate other than normal risks faced by mining projects in British Columbia in terms of environmental, permitting, taxation, socio economic, marketing, and political factors. The author is not aware of any legal or title issues that would materially affect the mineral resource estimate.

25.14.1.5 Mining

Pit walls failures are a common risk in open pit mines. Conservative pit slope designs have been adopted for the prefeasibility study; however, there is always the potential for zones of unstable and weak rock formations.

There are several risks that have been identified through the Study:

- Pit wall stability: there is a potential common risk of pit wall failure, which would affect health and safety as well as disrupt productivity with clean up in areas where the walls were not stable. If flatter pit wall slopes need to be established, it is expected that a higher strip ratio and reduction of mineral reserves could result.
- Geotechnical stability assessments: some of the mine infrastructure including the overburden stockpile and the operational ore stockpile have not undergone geotechnical stability assessments or local drilling. This should be conducted in a future project phase.
- Ore recovery or dilution: this can result in having operational results different than the design criteria, which would likely reduce mineral reserves.
- Mine design: there could be a failure to achieve production rates and to maintain design throughput. Future project phases will investigate this in more detail to alleviate this risk.

25.14.1.6 Project Infrastructure

25.14.1.6.1 Tailings Facility

Detailed site investigation of the dam foundations and laboratory strength testing of materials representative of the foundations has not been completed and will be required to confirm the proposed staged development of the tailings facility dams for verification of stability and to meet regulatory requirements. Glacial deposits as found in the Project area have the potential to be highly variable and it is possible that weak glacial sediments may be present in some areas.

The assumed 5 and 3 m average excavation depth beneath the North and West dams, respectively, is considered reasonable and practical. While the assumption has not been verified through site investigation, it is expected that this would be completed in the next project phase such that the assumption can be revisited and confirmed.

25.14.1.6.2 Construction

Constructability issues can lead to delays and cost overruns. A PFS-level constructability review was conducted to support this study and should be further advanced in the next project phase. Elements to be reviewed in further detail include traffic flow, construction timing (e.g., commencement of earthworks in summer periods, road upgrades, construction sequencing of major facilities) and more granular scheduling for commencement of early works (ie. Year -3). Further review of PFS staging and laydown allowances as well as consideration of a secondary access to the plant site during construction should also be made.

Winter weather can impact construction progress and maintenance during operation. Therefore, development of winterization plans for equipment, infrastructure and personnel safety is recommended.

25.14.1.6.3 Plant Site Geotechnical Conditions

At this stage of the study, there have been no geotechnical investigations conducted in the process plant area. As such, there is a risk to the estimated civil quantities, so it is important that they are verified during the next stage of the study through field investigations.

25.14.1.7 Metal Pricing and Payability

The ability of mining companies to fund the advancement of their projects through exploration and development is influenced by commodity prices. Although a marketing study was done to validate the pricing, variations in the commodity prices may lead to reduced or elevated revenues compared to those projected in this Study.

25.14.1.8 Environmental Studies, Permitting, and Social or Community Impact

A significant project risk is the potential for opposition to the Project with respect to cultural and environmental impacts. Attention to the quality and thoroughness of the baseline data collection, combined with meaningful engagement and consultation and strong technical work, and identification of avoidance and mitigation strategies, and compensation measures will be required to support the environmental assessment process.

An allowance has been included in the PFS cost estimate for cultural and environmental compensation mitigation projects to be completed in the first five years of operation (for example, fish habitat compensation). The number and scope of these compensation projects may exceed what was provided as an estimate in the PFS.

Reclamation cost estimates were developed for the PFS. A detailed evaluation of the strategies and costs may be higher than what was included in this estimate.

25.14.1.9 Refinery Option

The final location of the refinery has not yet been specified and will be highly dependent on the strategic participants in the Project. Considerations will be required to be made for waste handling, availability of utilities, contractors, work force, consumables and reagents supply, and concentrate transportation logistics when alternate locations are considered.

Refinery testwork completed is limited to batch testwork which was unable to capture the impact of various recycle streams on continuous operational performance. Continuous testwork is required in the next phase of study to verify the performance and specific equipment requirements for circuits with significant recycle streams. These are the counter-current leach, CoSX, and NiSX unit operation blocks.

25.14.2 Opportunities

25.14.2.1 Exploration

The Baptiste Deposit remains open at depth, with higher grade material at/below the base of the PFS open pit. Resource drilling has the potential to either increase grade or tonnage – there is approximately 500 Mt of resource below the PFS pit base averaging 0.145% DTR Ni, which is around 10% higher than the PFS average mill feed grade.

The PFS mine plan considers 44 Mt of Inferred material as waste. An infill drilling program conducted ahead of the feasibility study can be anticipated to convert this Inferred resource to Indicated, therefore they could be considered in the FS mine plan.

Two drilling seasons at the Van target have identified a potential deposit of comparable scale, grade, and geology to Baptiste. Further step-out drilling may see the Van target continue to grow, potentially to a comparable scale of the Baptiste Resource.

25.14.2.2 Metallurgy

Opportunities exist to optimize the grind-recovery relationship for nickel that has been developed in the recent testwork programs. This has the potential to reduce power requirements in comminution and benefit operating costs for the Project. Continued opportunities exist in terms of evaluating an iron concentrate being developed as a byproduct of nickel production.

25.14.2.3 Processing

The following opportunities are noted regarding future processing studies.

25.14.2.3.1 Flowsheet Development Optimization

With a deeper understanding of ore variability and the impacts on comminution and metallurgical performance, the flowsheet of the process plant may be optimized to further the overall project economics.

25.14.2.3.2 Magnetic Separation

Magnetic intensity data from testwork utilizing laboratory-scale equipment does not directly scale to industrial-scale equipment and there is an opportunity to decrease the magnetic intensity, and associated capital cost, with magnetic separation testing utilizing industrial-scale units.

Scavenger magnetic separation is currently considered for all the magnetic separation stages to align with the lab-scale test results. However, industrial-scale equipment could potentially achieve a similar performance without scavenger stages. This could have a potential impact on capital efficiency through removal of the respective scavenger circuit(s) if the industrial-scale equipment can be proven to outperform laboratory-scale equipment.

25.14.2.3.3 Awaruite Flotation

The awaruite flotation circuit design could be optimized with further testwork to improve the understanding of the awaruite flotation performance, including kinetics and nickel recovery. Improvement of nickel recovery to flotation concentrate would reduce nickel reporting to MHP, providing opportunities for selecting the best value recovery method.

25.14.2.3.4 Awaruite Tailings Leach

Awaruite tailings leach circuit design could be optimized with further testwork to improve nickel extraction rate and reduce acid consumption. This could further reduce the lime consumption at precipitation stages, providing opportunities for process operating cost savings.

25.14.2.3.5 Brine Purification and Precipitation

Testwork in the leachate purification and precipitation can verify the selected flowsheet, the precipitation conditions including pH and retention time, and assumptions used in selecting thickening and filtration equipment. This could provide opportunities to improve overall nickel recovery, MHP quality, and capital efficiency.

25.14.2.4 Mining

- Only Measured and Indicated Mineral Resources were considered for processing. Inferred Mineral Resources were treated as waste. Increasing Mineral Reserves through the incorporation of conversion of Inferred Mineral Resources into the mine plan has the potential to increase mine life and reduce strip ratios.
- Commodity prices may be higher than assumed, leading to enhanced project economics. The ability of the mine plan to adapt to higher commodity prices by altering cut-off policies and other strategic assumptions could potentially add value.
- At present, the overburden material is being categorized as standard waste. However, it is highly probable that a substantial portion of this material may not require drilling and blasting, especially when a more precise 'boundary' delineating overburden is established. Currently, a conservative strategy has been utilized, assuming all the material necessitates drilling and blasting. A potential optimization lies in revisiting this assumption when the understanding of the overburden boundaries and geomechanical characteristics are further refined.

25.14.2.5 Project Infrastructure

25.14.2.5.1 Tailings Facility

An opportunity for tailings facility optimization lies in evaluation of options to increase in-pit storage to reduce overall tailings facility embankment height in alignment with the Global Industry Standard for Tailings Management (GISTM) and philosophies of maximizing in-pit storage.

25.14.2.6 Environmental Studies, Permitting and Social or Community Impact

Tailings with a significant brucite content have the potential to sequester carbon dioxide. Further assessment of the value of carbon sequestration may provide economic benefits to the project.

Further geochemical testing and modelling with respect to seepage water collected during the closure period may allow for passive rather than active water treatment during the closure period. While this might not change the capital costs associated with constructing a water treatment system it would allow for significantly lower operating costs during this period.

25.14.2.7 Refinery

- Co-location of the refinery with a greater battery material hub presents process and logistic opportunities, such as elimination of the crystallization circuit.

- Sodium hydroxide has been selected as the primary neutralizing reagent and is a significant driver of operating cost. Correspondingly, the use of sodium hydroxide results in the formation of a sodium sulphate byproduct which has limited industrial uses and could be considered a waste product. The costs associated with both of these items could be significantly reduced through the use of any of the following;
    - Use of ammonia instead of sodium hydroxide resulting in lower reagent costs and the production of ammonium sulphate, a more readily saleable byproduct.
    - Use of lime instead of sodium hydroxide resulting in lower reagent costs. In this arrangement, ammonia would be used at the point of use and lime would be used to regenerate ammonium sulphate to ammonia through a lime-boil system resulting in the formation of a gypsum waste product.
    - Use of electrochemical salt splitting to convert sodium sulphate to sodium hydroxide and sulphuric acid, thereby creating a closed loop reagents system.
    - Eliminating of the NiSX unit operation block which is the dominant user of sodium hydroxide and a major user of sulphuric acid. Production of battery-grade nickel sulphate crystals via this route has not been proven and would require near-complete magnesium removal in CoSX and a nickel sulphate re-crystallization step for final purification of nickel sulphate from remaining element contained in the leach solution.

26 RECOMMENDATIONS

26.1 Path Forward – Budget and Schedule

The results presented in this technical report demonstrate that the Baptiste Nickel Project is technically viable and commercially positive. It is recommended that the project be advanced to the feasibility study (FS) phase.

A description of scope by function is then presented in the following subsections.

Table 26-1: Proposed Feasibility Study Budget

Function	Cost (US$M)
Geology and Resource	5.4
Mining	0.4
Metallurgy	2.3
Process & Infrastructure	6.4
Mine Waste and Water Management	4.5
Environment and Engagement	6.4
General	9.3
Total	**34.7**

26.2 Path Forward – Scope of Work

26.2.1 Geology and Resource

To support further development of the Baptiste Resource Estimate, conducting an infill resource drilling program targeting the conversion of resources from the Inferred category to the Indicated category is recommended. A program of approximately 8,075 m will support conversion of the bulk of Inferred resources from the PFS resource model, which total approximately 44 Mt. The recommended resource drilling program will commence late in the second quarter of 2024 and complete early in the fourth quarter of 2024.

In addition to the above-described resource drilling, the following drilling will also help inform updates to the Baptiste Resource Estimate and general geological understanding of the overall claims package:

- Condemnation drilling totalling 1,000 m in the plant site, tailings facility, and key infrastructure areas.
- Open pit geotechnical and hydrogeological drilling program.
- Geotechnical and hydrogeological drilling in the overall project area.

Results from the proposed feasibility resource drilling program will collectively inform updates to the Baptiste geological model. The updated Baptiste geological model will then serve as the basis for an updated Baptiste Resource Estimate.

A re-logging program of pre-2017 drill core is also proposed for the Baptiste Deposit. This re-logging program would allow for detailed alteration logging of drill core to consolidate different vintages of logging; it would also allow for integration of the alteration coding into the geological model.

The total cost of mentioned items is estimated at US$5.4M.

26.2.2 Mining

During the first half of 2024, is it recommended to complete several trade-off studies to define the opportunity to further increase project value, including:

- Decarbonization of pit haulage to reduce operating costs and carbon intensity, including evaluation of options such as trolley-assist, battery or hydrogen-powered haul trucks, and alternative fuel types.
- Evaluating autonomous haulage for potential to improve mine productivity and reducing on-site personnel requirements.
- Evaluation of semi-autonomous drills, reducing on-site personnel, eliminating exposure to potential hazards, and increase asset utilization as well as overall improved drilling and blasting performance (through increased accuracy).
- Optimization of mine planning parameters, including elements such as 400 t class haul trucks, 15 m bench heights, and a potential elevated cut-off grade in early mining years to increase mill feed grade.
- As part of the re-logging program recommended in the geology and resource section, the overburden thickness, boundaries and geomechanical characteristics can be re-evaluated. Note that the PFS adopted a conservative strategy, assuming that all the overburden material necessitates drilling and blasting.

In addition, conducting blasting simulations to refine the run-of-mine product size distribution assumption is recommended.

The total cost of mentioned items is estimated at US$0.4M.

26.2.3 Metallurgy

The metallurgical work outlined below is recommended for the next project phase.

Execution of pilot testwork to further demonstrate the metallurgical flowsheet at higher throughput and longer duration, including:

- Piloting of approximately 60-80 tonnes of bulk sample material.
- Piloting of approximately 3-6 tonnes of a new master composite sample.
- Further expansion to the comminution database through variability testwork on individual drill hole intervals and drill hole composites.

Further testwork to optimize the tailings leach circuit design, including:

- Leach optimization testwork, specifically focused on leaching conditions and acid consumption.
- Testwork validating the purification and precipitation unit operations.
- Dewatering testwork, focusing on concentrate, MHP, and tailings leach residue.
- Briquetting testwork to confirm briquetting criteria and product quality.
- Sulphide flotation testwork to determine the potential opportunity for nickel sulphide recovery.
- Iron ore testwork to determine the potential opportunity to produce a magnetite iron ore concentrate from tailings leach circuit residue.

In addition to the above test work which will support the PFS Base Case, additional hydrometallurgical test work is recommended to support further advancement of the Refinery Option (see Section 24). Conducting a larger-scale test work program on the awaruite concentrate generated from the pilot scale test work program above is recommended. This test work program will focus on demonstrating the leaching process at semi-pilot scale and demonstrating of the purification and crystallization unit operations at a larger bench scale.

The aforementioned metallurgical test work programs would be completed in the fourth quarter of 2024, ensuring that results are available and fully analyzed for inclusion in FS process design criteria.

The total cost of mentioned items is estimated at US$2.3M.

26.2.4 Process and Infrastructure

Completing several value engineering exercises is recommended for defining opportunity to further increase project value including:

- Trade-off study to evaluate acceleration of the metal production profile by means of executing the expansion to 162 kt/d earlier than Year 10, or a potential higher capacity initial build. This would include assessment of the effects on plant layout, camp facilities and infrastructure.
- Review and refinement of the grinding area of the concentrator, targeting an optimal blend of direct capital cost efficiency, constructability, operability and maintainability.
- Review and refinement of the overall construction execution approach to further refine the Project footprint (including temporary facility areas), execution approach, and overall execution duration.
- Trade-off study to evaluate the potential opportunity to produce a magnetite iron ore concentrate through upgrading of tails leach circuit residue.
- Trade-off study to evaluate a potential off-site administration center to support the Baptiste mine operations, to allow non-production roles to be located in a more urban setting.

The above value trade-off studies will be executed during the 2024 calendar year.

For the off-site power system, it is recommended design to advance to support formal entrance to the BC Hydro connection queue in 2024.

A formal logistics study is recommended for completion in 2024 to support the FS design basis, including identification of preferred shipping and other transit (road/rail) routes, maximum shipping envelopes, and maximum shipping weights.

In addition, select engineering and design elements will be advanced to support preparation of an Initial Project Description (IPD) in the middle of 2024, with subsequent EA support.

The total cost of mentioned items is estimated at US$6.4M.

26.2.5 Mine Waste and Water Management

The following is a list of recommendations for additional site investigations, testwork, and design studies to support FS design of the mine waste and water management facilities, which primarily includes the tailings facility:

- Continue early engagement with local communities and iterate tailings alternatives assessments accordingly with learnings from engagement and ongoing cultural and environmental baseline studies.
- Undertake detailed geotechnical and hydrogeological site investigation program(s) for the tailings facility dam foundations, water management pond foundations, other site infrastructure and open pit. A key requirement of the site investigation program(s) will be collection and laboratory testing of undisturbed samples of the tailings facility foundation materials.
- Complete detailed stability and seepage modelling of the proposed tailings facility dams based upon the results of the detailed site investigation program and laboratory testing of undisturbed samples.
- Complete a consolidation model to evaluate the change in density of the tailings mass throughout the mine life. The model will also provide an estimate of consolidation seepage (resulting from densification of the tailings mass) for use in an updated water balance model.
- Update the hydrometeorology inputs as further site-specific data is collected and develop a water balance model using a long-term synthetic climate and precipitation data set, to evaluate all years of mine operations and into closure.
- Optimize annual dam construction staging with overburden, Class A and Class B waste rock release from the mine plan. Investigate the possibility of advancing mining of additional Class A waste rock to offset Class B waste rock presently considered for use in the first five years of dam construction.
- Review and update the dam construction concept and proposed construction materials with the results of ongoing geochemical testing programs.
- Complete tailings material testwork as required to support FS design.

The total cost of mentioned items is estimated at US$4.5M.

26.2.6 Environment and Engagement

The following key activities are recommended to support ongoing project development and advancement to the next stage of engineering, and entrance into the environmental assessment process:

- Continued early engagement with local communities and Indigenous rightsholders.
- Cultural and environmental baseline data collection and reporting.
- Identifying and initiating additional environmental and engineering studies as required to support increased project definition and incorporate learnings from engagement and baseline programs.
- Formally entering into the EA process with provincial and federal agencies.

The total cost of mentioned items is estimated at US$6.4M.

26.2.7 General

The general cost center total is approximately $9.3M and includes camp costs, general site services, industry association fees, field work permits, safety programs, training, purchased equipment, owner's team, owner's team advisors/consultants, and miscellaneous expenses.

26.3 Refinery Option

The following are recommendations for the conceptual level Refinery Option:

- Metallurgy
  - Conduct continuous testing on the counter-current leach to assess circuit performance under steady-state conditions.
  - Confirm solid-liquid separation properties of the POX and atmospheric leach residues.
  - Optimize Cu cementation and Al/Cr removal circuit to minimize the associated nickel loss to these products.
  - Evaluate the use of use of different neutralizing reagents, producing different salt products (for example using ammonia and forming ammonium sulphate) to reduce operating costs.
  - Conduct continuous CoSX testing and/or circuit modelling to determine the required number of extract and scrub stages to achieve the required Co over Ni selectivity and minimize nickel co-extraction.
  - Conduct continuous NiSX testing and/or circuit modelling to determine the required number of extract and scrub stages to achieve the required Ni over Mg selectivity and maximize nickel co-extraction and to determine the required number of strip stages to maximize strip extent while producing strip solution with acceptable acidity to feed nickel sulphate crystallization.

- Conduct additional crystallization testing to define the required purge rates and maximum acceptable impurity and acidity in the mother liquor while maintaining acceptable product specifications.
- Determine the quality of the salt byproduct (sodium sulphate or other depending on selected neutralizing reagent).

- General
  - Through advanced discussions with potential strategic partners, identify:
    - A specific location for the off-site refinery.
    - Process simplification options, such as elimination of the nickel sulphate crystallization circuit.
    - Best value waste disposal strategy.
    - Potential market for refinery byproducts.

27 REFERENCES

Aeroquest, 2010, Report on a helicopter-borne tri-axial magnetic gradiometer survey: Internal report for Cliffs Natural Resources Exploration Inc., 27 p.

Allnorth Consultants Ltd., (2013a). Decar Nickel Property EBS Program Interim Quarterly Report September – November 2012. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.

Allnorth Consultants Ltd., (2013b). 2012 Meteorology Baseline Study Report. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.

Allnorth Consultants Ltd., (2013c). 2012 Hydrology Baseline Study Report. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.

Allnorth Consultants Ltd., (2013d). 2012 Surface Water Quality Baseline Study Report. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.

Allnorth Consultants Ltd., (2013e). 2012 Hydrogeology Baseline Study Report.

Allnorth Consultants Ltd., (2013f). 2012 Vegetation Baseline Study Report. Report prepared by Allnorth for Caracle Creek International Consulting Inc. and Cliffs Natural Resources.

Allnorth Consultants Ltd., (2013g). Soils Baseline Study Report. Report prepared by Allnorth for Caracle Creek International Consulting Inc. and Cliffs Natural Resources.

Allnorth Consultants Ltd., (2013h). 2012 Fish and Fish Habitat Baseline Study Report. Report prepared by Allnorth for Caracle Creek International Consulting Inc. and Cliffs Natural Resources.

Allnorth Consultants Ltd., (2013i). Aquatic Environment Baseline Study Report. Report prepared by Allnorth for Caracle Creek International Consulting Inc. and Cliffs Natural Resources.

Allnorth Consultants Ltd., (2013j). Summary of 2012 Archaeological Assessments. Report prepared by Allnorth for Caracle Creek International Consulting Inc. and Cliffs Natural Resources.

Allnorth Consultants Ltd., (2014a). 2012-2013 Terrestrial Wildlife Baseline Study Report. Report prepared by Allnorth for Caracle Creek International Consulting Inc. and Cliffs Natural Resources.

Allnorth Consultants Ltd., (2014b). 2013-2014 Meteorology Baseline Study Report. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.

Allnorth Consultants Ltd., (2014c). 2013-2014 Surface Water Quality Baseline Study Report. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.

Allnorth Consultants Ltd., (2014d). 2013-2014 Hydrogeology Baseline Study Report. Prepared for Caracle Creek International Consulting Inc. and Cliffs Natural Resources Exploration Canada Inc.

Armstrong, J.E., 1949, Fort St. James map-area, Cassiar and Coast Districts, British Columbia: Geological Survey of Canada Memoir 252, Geological Survey of Canada, 210 p.

BC Hydro, (2023). FPX Nickel-Baptiste Load Interconnection – Feasibility Study

Boudier, F., and Nicolas, A., 1985, Harzburgite and lherzolite subtypes in ophiolitic and oceanic environments: Earth and Planetary Science Letters, v. 76, nos. 1–2, p. 84–92.

British Columbia Ministry of Environment (BCMoE), (2016). Water and Air Baseline Monitoring Guidance Document for Mine Proponents and Operators-Version 2.

Britten, R., 2009, Field season 2008, geology and geochemistry, Decar property, BC: BCMEM assessment report 30793 ARIS 30793, 70 p.

Britten, R., 2010, Field season 2009, geology and geochemistry, Decar property, BC: BCMEM assessment report 31334 ARIS 31334, 51 p.

Britten, R., 2016, Regional metallogeny and genesis of a new deposit type - disseminated awaruite (Ni3Fe) mineralization hosted in the Cache Creek Terrane: Economic Geology, v. 112, no. 3, p. 517–550.

Britten, R., and Rabb, T., 2010, Field Season 2009 geology and geochemistry, Decar Property, BC: BC Ministry of Energy, Mines and Petroleum Resources BCMEM assessment report 31551 ARIS 31551, 54 p., accessed at https://aris.empr.gov.bc.ca/ArisReports/31551.PDF.

Britten, R., and Rabb, T., 2011, Field season 2010, airborne gradient magnetic and IP geophysical surveys, Decar property, BC: BCMEM assessment report 31999 ARIS 31999, 94 p.

Caracle Creek International Consulting Inc., (2012). ARD and ML Assessment Report.

Carr, I., 2022, 2021 drilling, geological, and geophysical report on the Decar nickel-iron alloy project: Unpublished assessment report prepared for FPX ARIS TBD, 686 p.

Centerra Gold, 2018, Endako Mine history, https://www.centerragold.com/operations/molybdenum-business-unit/ (https://www.centerragold.com/operations/molybdenum-business-unit/).

DGI, 2012a, Decar property, 2-4C summary: Internal report for Cliffs Natural Resources Ltd, 6 p.

DGI, 2012b, Logistics and final report: 2011 Decar property physical properties, directional and, optical and acoustic televiewer survey: Internal report for Cliffs Natural Resources, 17 p.

Ditson, G., Johnson, T., Jakubowski, W., and Yeager, D., 2008, Assessment report on geochemical work performed on the POLYMAC property: BCMEM assessment report 29697 ARIS 29697, 62 p.

Ecofor, (2022). Preliminary Field Reconnaissance of FPX Nickel's Decar Property Footprint.

Environment Canada, 2022, Canadian climate normals 1981-2010 station data, Babine Lake Pinkut Creek, British Columbia, Retrieved 01 August 2022 (https://climate.weather.gc.ca/climate_normals/results_1981_2010.

ERM (2021). Decar Project – Fish Community and Habitat Reconnaissance Trip Report, September 2021.

ERM, (2023a). Baptiste Nickel Project – 2022 Meteorology Baseline Report.

ERM, (2023b). Baptiste Nickel Project – 2022 Hydrology Baseline Report.

ERM, (2023c). Baptiste Nickel Project – 2022 Surface Water Quality Baseline Report.

Fitzgerald, J., Jago, C., Jankovic, S., Simonian, B., Taylor, C., and Borntraeger, B., 2019, Technical Report on the Mount Milligan Mine North-Central British Columbia: NI 43-101 Technical Report for Centerra Gold Corp.

Forbes, J.R., 1988, Geological and Geochemical Report on the New Claim Group: BCMEM assessment report 17944 ARIS 17944, 36 p.

FPX Nickel, 2014, First Point Minerals provides Decar update, News release issued by FPX Nickel, 27 August 2014.

FPX Nickel, 2015, First Point Minerals to re-establish 100% ownership of Decar Project, News release issued by FPX Nickel, 08 September 2015.

Frost, R.B., 1985, On the stability of sulfides, oxides and native metals in serpentinite: Journal of Petrology, v. 26, p. 31–63.

Gabrielse, H., 1985, Major dextral transcurrent displacements along the northern Rocky Mountain trench and related lineaments in north-central British Columbia: Geological Society of America Bulletin, v. 96, no. 1, p. 1–14.

Guinet, V., 1980, Prospecting report on C.R. claims: BCMEM assessment report 8135 ARIS 8135, 12 p.

Hanson, Jeremy, 2023. Technical report on the MAC project, Omineca Mining Region, British Columbia, Canada. Technical report prepared for Transforma Resources Corp., 69 pp.

Klohn Crippen Berger (KCB), (2012). Decar Nickel Project – Preliminary Environmental Baseline Report.

Knight Piésold Ltd. (KP, 2023). Baptiste Nickel Project Tailings Facility Preliminary Design Report. Ref. No. VA101-824/6-2.

Le Couteur, P.C., 2008, Report on Ni-Fe alloys in 13 samples: Internal report for First Point Minerals Corp, 19 p.

Logan, J.M., Schiarizza, P., Struick, L.C., Barnett, C., Nelson, J.L., Kowalczyk, P., Ferri, F., Mihalynuk, M.G., Thomas, M.D., Gammon, P., Lett, R.E., Jackaman, W., and Ferbey, T., 2010, Bedrock geology of the QUEST map area, central British Columbia: Geological Survey of Canada Open File 6476.

McLaughlin, M., Ronacher, E., Palich, J., Baker, J., Ni, W., Kanhai, T., Kesavanathan, D., Dejan, V., Jansons, K., and Carter, A., 2013, Preliminary Economic Assessment Decar Nickel Project, British Columbia, Canada (Amended): Preliminary Economic Assessment Report, 504 p.

Monger, J.W.H., Wheeler, J.O., Tipper, H.W., Gabrielse, H., Harms, T.A., Struik, L.C., Campbell, R.B., Dodds, C.J., Gehrels, G.E., and O'Brien, J., 1991, Part B, Cordilleran terranes, in Gabrielse, H. and Yorath, C.J. eds., Geology of Cordilleran Orogen in Canada: Geological Survey of Canada, p. 281–327.

Mowat, U., 1988a, Geochemical sampling on the Van group, Klone group, Mid claim: BCMEM assessment report 17173 ARIS 17173, 172 p.

Mowat, U., 1988b, Geochemical sampling, prospecting and mapping on the Van group, Klone group and Mid claim: BCMEM assessment report 18089 ARIS 18089, 172 p.

Mowat, U., 1990, Mapping and drilling program on the Mount Sydney Williams property: BCMEM assessment report 20541 ARIS 20541, 94 p.

Mowat, U., 1991, Drilling program on the Mount Sydney Williams property: BCMEM assessment report 21870 ARIS 21870, 73 p.

Mowat, U., 1994, Drilling program on the Mount Sidney Williams Gold property: BCMEM assessment report 23569 ARIS 23569, 94 p.

Mowat, U., 1997a, A geochemical/petrographic report on the Mount Sidney Williams property: BCMEM assessment report 24906 ARIS 24906, 46 p.

Mowat, U., 1997b, Geochemical and petrographic report on the Star claims: BCMEM assessment report 25002 ARIS 25002, 50 p.

Mowat, U., 1997c, Sampling and metallurgical report on the Mount Sydney Williams property: BCMEM assessment report 25278 ARIS 25278, 162 p.

Mowat, U., 1998, Mapping and sampling on the Mount Sydney Williams property: BCMEM assessment report 25727 ARIS 25727, 57 p.

Mowat, U., 1999, Mapping and sampling on the Mount Sydney Williams property: BCMEM assessment report 26062 ARIS 26062, 45 p.

Mowat, U., 2000, Compilation and sampling on the Fran claims: BCMEM assessment report 26282 ARIS 26282, 48 p.

Mowat, U., 2001, Mapping and sampling on the Mount Sydney Williams property: BCMEM assessment report 26445 ARIS 26445, 31 p.

Mowat, U., 2002, Sampling on the Mount Sydney Williams property: BCMEM assessment report 26993 ARIS 26993, 34 p.

Mowat, U., 2004, Sampling on the Mid and Klone 7 claims: BCMEM assessment report 27375 ARIS 27375, 25 p.

Mowat, U., 2005, Sampling on the One-eye 1 and Klone 1 claims: BCMEM assessment report 27605 ARIS 27605, 27 p.

Mowat, U., 2007, Sampling and grid preparation on the Klone 5 and Klone 6 claims: BCMEM assessment report 28806 ARIS 28806, 33 p.

Onsite Engineering, (2023). Pre-Feasibility Offsite Access Roads Summary Report & Appendices

Palich, J., and Qian, W., 2012, Earthprobe survey interpretation report, Decar nickel property, Mt Sidney Williams, British Columbia: BCMEM assessment report 33135 ARIS 33135, 70 p.

Pezzot, E.T., and Vincent, J.S., 1982, Geophysical report on an airborne VLF-EM and magnetometer survey, CR1-CR 6 claims: BCMEM assessment report 10286 ARIS 10286, 29 p.

Plouffe, A., 2000, Surficial geology, Fort Fraser, British Columbia.

Rabb, T., and Carr, I., 2022, 2021 Decar mapping program: property geology: Internal report, 24 p.

Ronacher, E., 2013, Assessment report - LiDAR survey: BCMEM assessment report 33953 ARIS 33953, 63 p.

Ronacher, E., Harnois, L., and Baker, J., 2012a, Assessment report, Decar Nickel Property, British Columbia, Canada: BCMEM assessment report 33414 ARIS 33414, 554 p.

Ronacher, E., Baker, J., Palich, J., Broad, P., and Thesen, C., 2012b, Independent technical report, Decar Nickel Property, British Columbia, Canada: NI43-101 Technical Report for First Point Minerals Corp., 156 p.

Ronacher, E., Baker, J., Palich, J., Broad, P., and Thesen, C., 2013, Resource update - Decar Nickel Property, British Columbia, Canada: NI43-101 Technical Report prepared for First Point Minerals Corp., 160 p.

Schiarizza, P., and MacIntyre, D.M., 1999, Geology of the Babine Lake - Takla Lake area, central British Columbia (93K/11, 12, 13, 14; 93N/3, 4, 5, 6): Geological Fieldwork 1998, British Columbia Ministry of Energy and Mines, British Columbia Geological Survey Paper 1999-1, p. 33–68.

Shas-Ti Environmental, (2023a). 2022 Decar Aquatics Summary Report.

Shas-Ti Environmental, (2023b). 2022 Decar Fisheries Summary Report.

Shas-Ti Environmental, (2023c). 2022 Decar Vegetation Data Summary Report.

Shas-Ti Environmental, (2023d). 2022 Decar Wildlife Data Summary Report.

Staples, L.P., Bowen, J.M., Bemier, S.B., Warren, D.A., Scott, C.C., Duncan, J.F., Murphy, B.A., Bertrand, V.J., Scott, K.C., and Latulippe, S., 2013, Technical Report on the Dumont Ni project, Launay and Trecesson townships, Quebec, Canada: Royal Nickel Corporation NI 43-101 Technical Report for Royal Nickel Corp.

Steinthorsdottir, K., 2021, Formation and preservation of brucite and awaruite in a serpentinized ultramafite, central British Columbia: implications for carbon sequestration and nickel recovery. M.Sc thesis: University of British Columbia: 383 p.

Stelling, D., 1975, A rock sampling and prospecting report on the Pauline mineral claims: BCMEM assessment report 5648 ARIS 5648.

Struik, L.C., Schiarizza, P., Orchard, M.J., Cordey, F., Sano, H., MacIntyre, D.G., Lapierre, H., and Tardy, M., 2001, Imbricate architecture of the upper Paleozoic to Jurassic oceanic Cache Creek terrane, central British Columbia: Canadian Journal of Earth Sciences, v. 38, p. 495–514.

Svoboda, J., 2004, Magnetic techniques for the treatment of materials: Springer Science & Business Media, 642 p.

Terra Remote Sensing Inc, 2013, Aerial survey - LiDAR, Central BC: Data acquisition report prepared for Caracle Creek International Consulting, 15 p.

Vanderzee, S.S.S., Power, I.M., Dipple, G.M., and Bradshaw, P.M.D., 2018, Carbon mineralization in ultramafic tailings, central British Columbia: a prospect for stabilizing mine waste and reducing greenhouse gas emissions, in Geoscience BC Summary of Activities 2017: Minerals and Mining, Geoscience BC, Report 2018-1: p. 109–112.

Voormeij, D.A., and Bradshaw, P., 2008, Summer 2007 geochemical analysis on rock samples, Decar Property, BC: BCMEM assessment report 30499 ARIS 30499, 25 p.

Walcott, P.E., 2011, A logistical report on magnetic & induced polarization surveying, Decar property, Trembleur Lake area: Internal report for First Point Minerals Corp, 39 p.

Whittaker, P.J., 1983, Geology and petrogenesis of chromite and chrome spinel in the alpine-type peridotites of the Cache Creek Group, British Columbia. Doctoral thesis: Carleton University: 339 p.

Appendix A – MINERAL CLAIMS

Tenure Number	Claim Name	Owner	Issue Date	Good To Date	Area (ha)	Overlap Description
559615	WILL 1	FPX (100%)	31-May-07	14-Nov-33	464.76	Legacy claim
559616	WILL 2	FPX (100%)	31-May-07	14-Nov-33	464.76	Legacy claim
559617	WILL 3	FPX (100%)	31-May-07	14-Nov-33	464.76	
559618	WILL 4	FPX (100%)	31-May-07	14-Nov-33	446.35	
575674	WILL 5	FPX (100%)	08-Feb-08	14-Nov-33	446.49	
575675	WILL 6	FPX (100%)	08-Feb-08	14-Nov-33	446.63	
575677	WILL 7	FPX (100%)	08-Feb-08	14-Nov-33	465.19	
575678	WILL 8	FPX (100%)	08-Feb-08	14-Nov-33	464.95	
575679	WILL 9	FPX (100%)	08-Feb-08	14-Nov-33	464.72	
575680	WILL 10	FPX (100%)	08-Feb-08	14-Nov-33	465.19	
575681	WILL 11	FPX (100%)	08-Feb-08	14-Nov-33	446.38	
575682	WILL 12	FPX (100%)	08-Feb-08	14-Nov-33	297.65	Legacy claim
575683	WILL 13	FPX (100%)	08-Feb-08	14-Nov-33	390.4	
575684	WILL 14	FPX (100%)	08-Feb-08	14-Nov-33	223.37	
575686	WILL 15	FPX (100%)	08-Feb-08	14-Nov-33	316.24	
594247	BAP 1	FPX (100%)	14-Nov-08	14-Nov-33	446.78	
594248	BAP 2	FPX (100%)	14-Nov-08	14-Nov-33	335.14	
594249	BAP 3	FPX (100%)	14-Nov-08	14-Nov-33	465.43	
594250	BAP 4	FPX (100%)	14-Nov-08	14-Nov-33	446.7	
594251	BAP 5	FPX (100%)	14-Nov-08	14-Nov-33	390.88	
594252	KAR 1	FPX (100%)	14-Nov-08	14-Nov-33	464.53	
594254	KAR 2	FPX (100%)	14-Nov-08	14-Nov-33	464.29	
594255	KAR 3	FPX (100%)	14-Nov-08	14-Nov-33	464.29	Mineral reserve, district lot
594256	KAR 4	FPX (100%)	14-Nov-08	14-Nov-33	427.27	Mineral reserve, district lot
594257	KAR 5	FPX (100%)	14-Nov-08	14-Nov-33	371.63	Mineral reserve, district lot
594258		FPX (100%)	14-Nov-08	14-Nov-33	464.52	
594259	KAR 7	FPX (100%)	14-Nov-08	14-Nov-33	445.97	Legacy claim
594260	KAR 8	FPX (100%)	14-Nov-08	14-Nov-33	297.19	
594262	KAR 9	FPX (100%)	14-Nov-08	14-Nov-33	408.72	
594263	KAR 10	FPX (100%)	14-Nov-08	14-Nov-33	389.92	
602564		FPX (100%)	14-Apr-09	14-Nov-33	18.58	
602566		FPX (100%)	14-Apr-09	14-Nov-33	148.66	
603803	VAN 1	FPX (100%)	03-May-09	14-Nov-33	464.51	District lot
669586	BAP 6	FPX (100%)	16-Nov-09	14-Nov-33	260.51	Legacy claim
669625	BAP 7	FPX (100%)	16-Nov-09	14-Nov-33	446.96	
669645	BAP 8	FPX (100%)	16-Nov-09	14-Nov-33	446.94	

Tenure Number	Claim Name	Owner	Issue Date	Good To Date	Area (ha)	Overlap Description
669665	BAP 9	FPX (100%)	16-Nov-09	14-Nov-33	446.91	
839601	MID 1	FPX (100%)	03-Dec-10	03-Dec-33	74.4	
839604	MID 2	FPX (100%)	03-Dec-10	03-Dec-33	446.66	District lot
839607	MID 3	FPX (100%)	03-Dec-10	03-Dec-33	427.88	District lot
839610	MID 4	FPX (100%)	03-Dec-10	03-Dec-33	465.28	Mineral reserve, district lot
839615	MID 5	FPX (100%)	03-Dec-10	03-Dec-33	427.9	Mineral reserve, district lot
839617	MID 6	FPX (100%)	03-Dec-10	03-Dec-33	464.9	Mineral reserve, district lot
839618	MID 7	FPX (100%)	03-Dec-10	03-Dec-33	464.75	Mineral reserve, district lot
839620	MID 8	FPX (100%)	03-Dec-10	03-Dec-33	427.39	Mineral reserve, district lot
839621	MID 9	FPX (100%)	03-Dec-10	03-Dec-33	464.33	Mineral reserve, district lot
839622	MID 10	FPX (100%)	03-Dec-10	03-Dec-33	148.55	Mineral reserve, district lot
895893	NEY 1	FPX (100%)	02-Sep-11	02-Sep-33	446.56	Legacy claim
895899	NEY 2	FPX (100%)	02-Sep-11	02-Sep-33	465.29	Legacy claim
895902	NEY 3	FPX (100%)	02-Sep-11	02-Sep-33	446.92	
895904	NEY 4	FPX (100%)	02-Sep-11	02-Sep-33	465.52	
895905	NEY 5	FPX (100%)	02-Sep-11	02-Sep-33	390.91	
895907	NEY 6	FPX (100%)	02-Sep-11	02-Sep-33	465.54	
895909	NEY 7	FPX (100%)	02-Sep-11	02-Sep-33	447.11	Provincial Park
895910	NEY 8	FPX (100%)	02-Sep-11	02-Sep-33	447.16	Provincial Park
895911	NEY 9	FPX (100%)	02-Sep-11	02-Sep-33	465.76	Provincial Park
895912	NEY 10	FPX (100%)	02-Sep-11	02-Sep-33	465.74	Provincial Park
895913	NEY 11	FPX (100%)	02-Sep-11	02-Sep-33	335.31	Provincial Park
895914	NEY 12	FPX (100%)	02-Sep-11	02-Sep-33	446.56	
1013225	BT GROUP	FPX (100%)	26-Sep-12	02-Sep-33	632.34	District lot
1053094	BT GROUP	FPX (100%)	12-Jul-17	12-Jul-32	111.63	Protected
1069701	NEY13	FPX (100%)	16-Jul-19	12-Jul-32	111.67	Protected